E LIBRARY

August 24, 1929

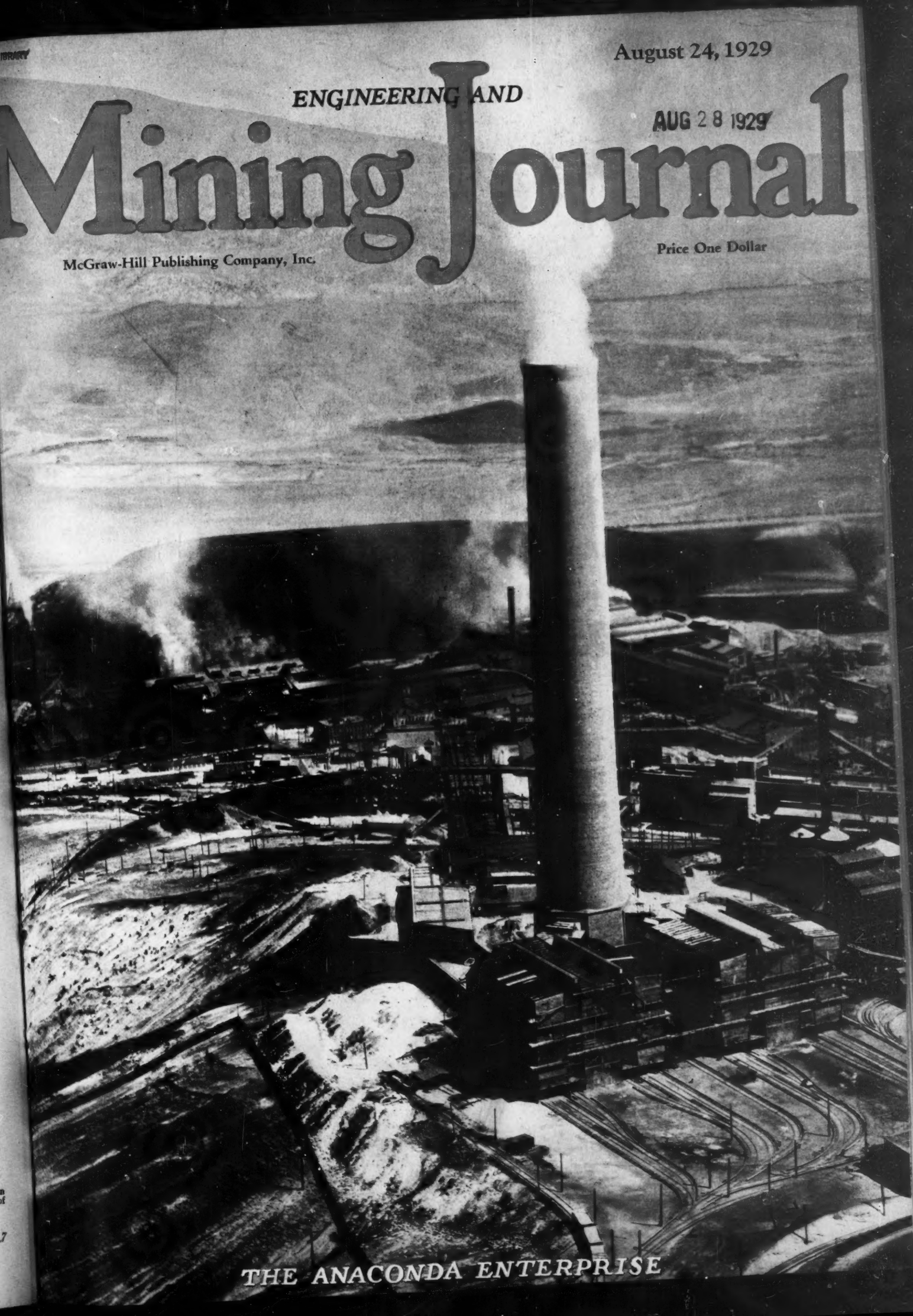
ENGINEERING AND

# Mining Journal

AUG 28 1929

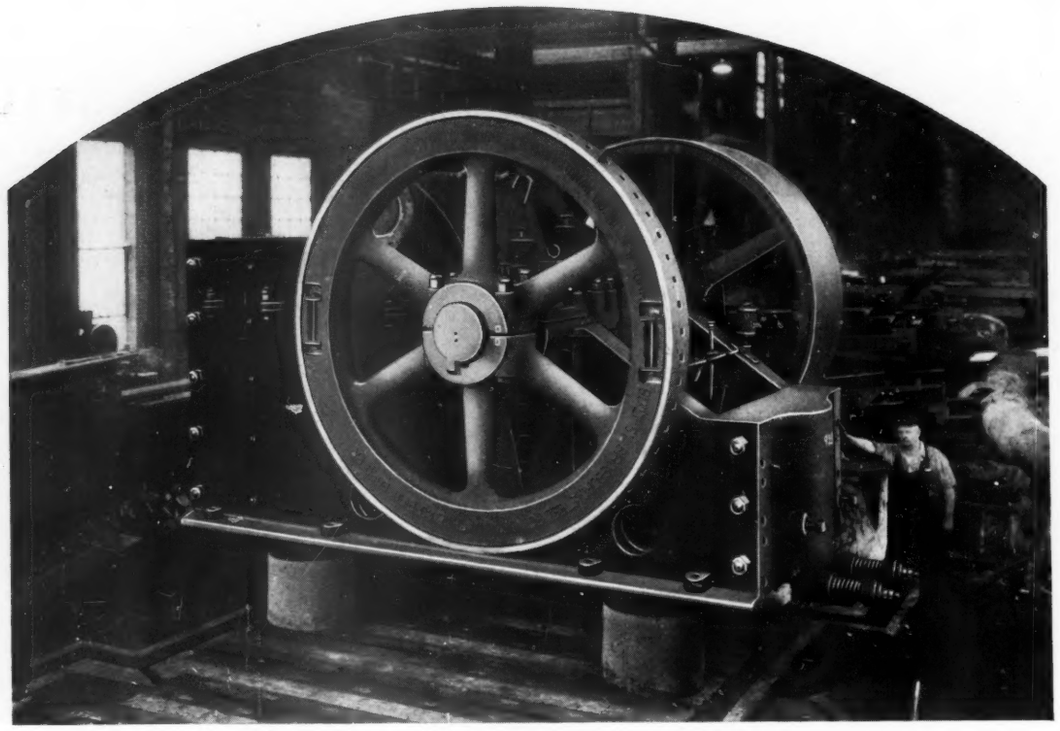
McGraw-Hill Publishing Company, Inc.

Price One Dollar



THE ANACONDA ENTERPRISE

128, No.7



Photograph of the 48"x60" BUCHANAN Type "C" Jaw Crusher in our shops before shipping to the Hollinger Mines at Timmins, Ontario

# Two Million Tons of Tough Quartz Annually Crushed Underground

This 48 in. x 60 in. BUCHANAN was installed in 1924 by the Hollinger Consolidated Gold Mines, Ltd., of Timmins, Ontario, Canada. The first year it handled 2,000,000 tons, crushing run-of-mine material to 6 in. size. Two years later, to care for increased capacity, a second BUCHANAN of the same size was installed at a lower level. Since installation both crushers have operated without a forced shutdown.

The service these crushers are giving Hollinger is a typical example. Many other prominent mining companies have assured themselves of continuous and efficient production by installing BUCHANANS in their underground crusher stations.

*Some representative users of Type C sectionalized for underground service:*

| | |
|---|---|
| Hollinger Consolidated Gold Mines, Ltd., Timmins, Ontario | 2—48"x60's. |
| Dome Mines Ltd., South Porcupine, Ontario | 2—36"x54's. |
| Orkla-Grube-Aktiebolag, Lokens, Verk, Norway | 2—36"x42's. |
| Porcupine Paymaster Mines, Ltd., South Porcupine, Ontario | 1—30"x42". |
| Replogle Mine, Wharton, New Jersey | 1—30"x42". |
| Ketsof Mining Company, Retsof, New York | 1—30"x42". |

BUCHANAN Type "C" Crushers are built with jaw openings ranging from 24 in. x 36 in. to 54 in. x 72 in. All sizes can be sectionalized for underground installation.

*Write for Bulletin No. 110.*

# BUCHANAN

**C. G. BUCHANAN COMPANY, Inc.**
**90 WEST STREET, NEW YORK, N. Y.**

**CRUSHING MACHINERY DEPARTMENT OF BIRDSBORO STEEL FOUNDRY & MACHINE CO.**

ENGINEERING AND MINING JOURNAL. August 24, 1929. Vol. 128, No. 8. Published weekly. McGraw-Hill Publishing Company, Inc., Tenth Avenue at 36th Street, New York, N. Y. United States, $6 a year; 25 cents a copy. U. S. Possessions, Canada, Mexico and the countries of Central and South America, $7.50 a year. All other foreign countries, $10 a year. Entered as second-class matter, June 20, 1879, at the Post Office, New York, N. Y., under the Act of March 3, 1879. Printed in U. S. A.

*Volume 128*

# *ENGINEERING AND* MINING JOURNAL

*Number 8*

*A consolidation of "Engineering and Mining Journal" and "Mining and Scientific Press"*

A. W. ALLEN, *Editor*

*Associate Editors:* E. H. ROBIE, A. H. HUBBELL, New York. GEORGE J. YOUNG, San Francisco
*Assistant Editors:* W. N. P. REED and H. D. KEISER, New York. *Market Editor:* H. H. WANDERS. *Publishing Director:* H. W. CLARKE

## Contents

*Copyright, 1929, by McGraw-Hill Publishing Company, Inc.*

**August 24, 1929**

### *The Anaconda Enterprise*

**Editorial—Meeting the Test of Large Responsibility** 275
By A. W. Allen

**Among the Directing Executives at Anaconda Headquarters** 276
Illustrations

**Ore Discovery and Development of Fundamental Importance** 277
By Reno H. Sales

**Evolution of Mining Practice at Butte** 280
By William B. Daly

**Timber and Lumber Interests** 285
By W. C. Lubrecht

**Modern Metallurgical Miracles** 287
By Frederick Laist

**Selective Flotation of Lead-Zinc Ores at Tooele, Utah** 291
By A. B. Young and W. J. McKenna

**Both Copper and Zinc Ores Treated by Selective Flotation** 295
By Bayard S. Morrow

**Development of Copper Smelting at Anaconda** 301
By Louis V. Bender

**Promoting Sales of the Product** 305
By J. N. McDonald

**Refining Anaconda Copper at Raritan and Great Falls** 306
By W. T. Burns

**Phosphates and Sulphuric Acid** 312
By E. L. Larison

**Smelting Lead at Tooele** 313
By J. O. Elton and B. L. Sackett

**Western Lead Refined at East Chicago by Parkes Process** 316
By G. E. Johnson

**Pioneer Work in Development of White Lead Manufacture by Sperry Electrolytic Process** 318
By R. G. Bowman

**Electrolytic Zinc Practice at Great Falls and Anaconda** 319
By Albert E. Wiggin and Russel B. Caples

**Fuming Off Zinc From Lead Slags at East Helena** 325
By Alexander Laist

**Arsenicals—An Important By-product** 325
By H. C. Gardiner

**French Process Modernized by Anaconda in Production of Zinc Oxide at East Chicago** 326
By F. O. Case

**Mining the World's Largest Copper Ore Deposit at Chuquicamata, Chile** 327
By H. C. Bellinger

**Manufacturing the Raw Material** 330
By John A. Coe

**Leaching, Flotation, Smelting Included in Andes' Operations at Potrerillos, Chile** 332
By William Wraith

**The Polish Enterprise** 333
By George Sage Brooks

**Anaconda Wire & Cable Company One of the Newest Subsidiaries** 335
By H. Donn Keresey

**How Anaconda Applies Electricity in Mining and Metallurgy** 336
By C. D. Woodward

**Half a Million Tons of Coal Annually Produced by Anaconda** 339
By F. W. C. Whyte

**Industrial Relations and Safety Work** 340
By John L. Boardman

---

**The Market Report** 341

**Service Directory** 195

**Searchlight Section** 196-199

**What and Where to Buy** 200, 202, 204

**Advertising Index** 206

**McGRAW-HILL PUBLISHING COMPANY, INC., Tenth Ave. at 36th St., New York, N. Y.**

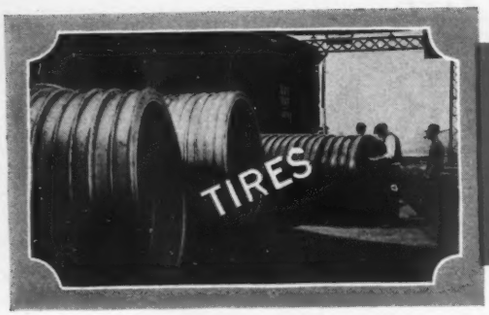
JAMES H. McGRAW, Chairman of the Board
MALCOLM MUIR, President
JAMES H. McGRAW, Jr., Vice-Pres. and Treas.
EDWARD J. MEHREN, Vice-President
MASON BRITTON, Vice-President
EDGAR KOBAK, Vice-President
HAROLD W. McGRAW, Vice-President
H. C. PARMELEE, Editorial Director
C. H. THOMPSON, Secretary

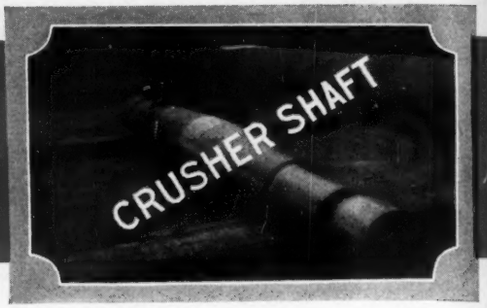
*Member, Audit Bureau of Circulations, and Associated Business Papers, Inc.*

NEW YORK District Office, 285 Madison Ave.
WASHINGTON, National Press Building
CHICAGO, 520 North Michigan Avenue
PHILADELPHIA, 1600 Arch St.
CLEVELAND, Guardian Building
ST. LOUIS, Bell Telephone Building
SAN FRANCISCO, 883 Mission St.
BOSTON, 1427 Statler Bldg.
GREENVILLE, S. C., 1301 Woodside Bldg.
DETROIT, 4-257 General Motors Building
LONDON, 6 Bouverie St., London, E. C. 4.

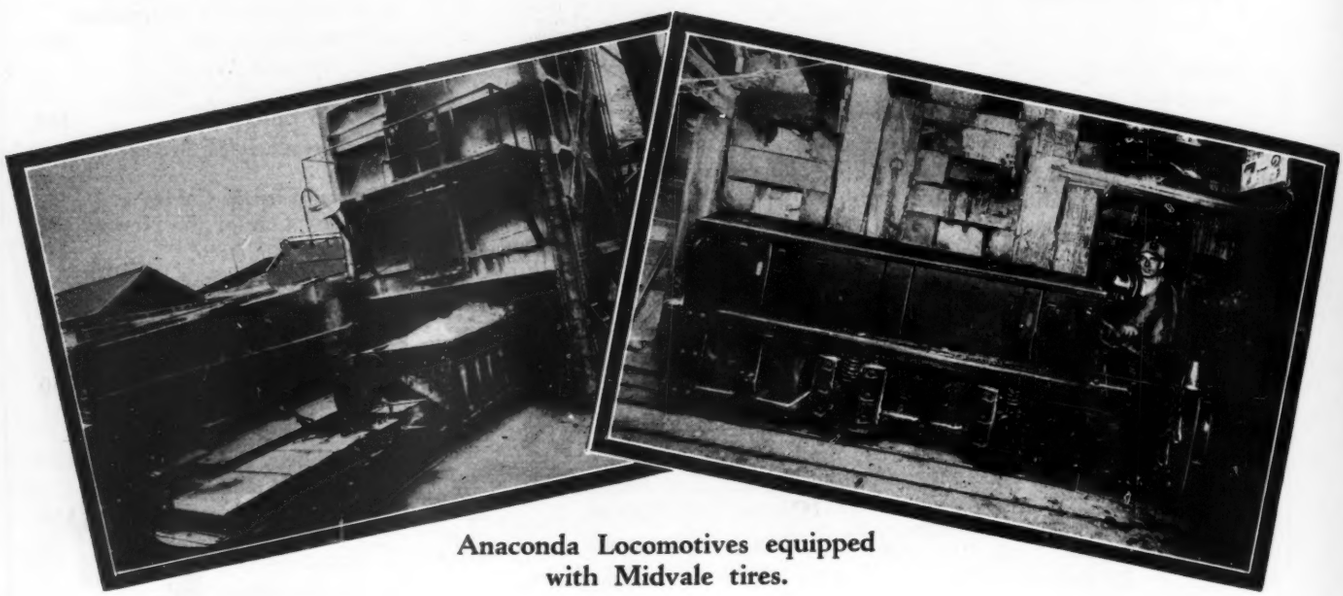
**12,000 Copies Printed This Issue**

**Total Number of Issues to Date 3,296**






# When Anaconda



**Anaconda Locomotives equipped with Midvale tires.**

## Midvale Roll Shells

**give highly satisfactory service. They are hard enough to withstand the wear and abrasion incident to crushing hard ores and tough enough to avoid breakage while shrunk or keyed to arbors. They wear down to a fraction of an inch in thickness without breaking.**

## Midvale Crusher Shafts

**are made in two compositions—ordinary carbon or a special alloy. Our special alloy withstands shock and fatigue to a remarkable degree and is practically unbreakable.**

MID

*ENGINEERING AND*
# MINING JOURNAL

A McGRAW-HILL PUBLICATION — A. W. ALLEN, *Editor* — ESTABLISHED 1866

*New York, August 24, 1929*

VOLUME 128 . . . . NUMBER 8

## Meeting the Test of Large Responsibility

ATTAINMENT of mere bigness is not in itself worthy of special commendation; but as connoting a co-ordination of interrelated activities and allied interests, whereby costs are reduced, market opportunities developed, and sufficient financial reserve is secured to resist successfully the effect of temporary depression and to permit continuous research in an effort to improve working conditions—as such, bigness is to be welcomed as beneficial to all. . . . The great industrial organization of today stands at the bar of public opinion. It must justify its size, its very existence, and meet a type of criticism from which the small company is exempt. It is expected to blaze a pioneer trail leading to efficient operating technique, sound ethical practice, consideration for the rights of employees and competitors alike, coupled with tact and a sense of fairness that prompt adequate recognition of the work of others—of that army of inventors who indirectly contribute to all great achievements. . . . From the spotlight scrutiny of an examination to determine industrial quality, the Anaconda Copper Mining Company emerges as one of the world's most progressive and reputable business organizations, one that gives evidence of a commercial and professional statesmanship in the conduct of its operations that has brought excellent financial returns to its stockholders, and has, in addition, contributed in no small measure to the stability and prosperity of the mining industry as a whole, and to the well-being of the countries in which it operates. An institution of enviable standing, of wide scope and influence, it has, moreover, pioneered in penetrating beyond the usual confines of the production field—even to the fabricating and selling of the products and by-products of its mines and mills and smelters. . . . From Anaconda's Montana and overseas properties, where operations are directed and performed with a skill that has evoked the admiration of visitors of all nationalities, has come valuable and needed inspiration in matters humanitarian as well as technical. The company, proud of the reasons for progress, has nothing to hide, and *Engineering and Mining Journal* feels privileged to act as a clearing house for the valuable information to be found in the pages of this issue, the release of which offers further evidence of a long-established policy of technical co-operation with other elements in the industry for the advancement of mining and metallurgical practice. . . . The executives and engineers and technicians of the Anaconda company and its subsidiaries have given freely. So generous a gesture to the mining industries of the world compels response; it cannot fail to enhance company morale and to encourage an international freemasonry of ideas and experiences that will demolish barriers—geographical, social, intellectual—that hinder progress and restrict prosperity.

> **"The motives influencing the great body of writers . . . to give to the brethren of their craft the results of their own often dearly earned experience are various and complicated. But, in the majority of cases, the impulse originates in the desire for reciprocity, and in the hope that others will tell what they know, in return for what we ourselves communicate, and that therefore we shall learn at least as much as we can teach."—*From the Presidential address of Dr. James Douglas before the American Institute of Mining Engineers, 1899.***

A. W. Allen

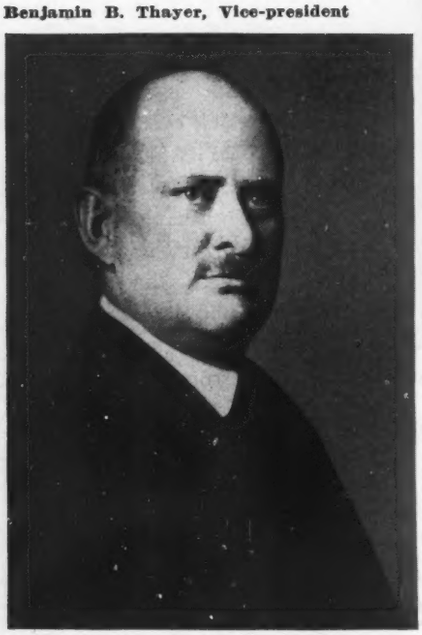
# Among the Directing Executives at Anaconda Headquarters

Cornelius F. Kelley
President

John D. Ryan
Chairman of the Board

James R. Hobbins, Vice-president

Benjamin B. Thayer, Vice-president



Robert E. Dwyer, Vice-president

A part of "the richest hill on earth"

# Ore Discovery and Development of Fundamental Importance

## *How Anaconda's Large Geological Staff Is Employed*

By RENO H. SALES
*Chief Geologist*

USUALLY the earnings of mining companies are made wholly from ore; and few mines, if any, have unlimited ore reserves. The mine operator therefore deals with a wasting asset. The main objective of the miner is to secure a maximum profit from ore contained in a given tract of mining ground. This objective can be reached only by making reasonably sure that all profitable ore existing in the ground is found and that the costs of finding and extracting the ore are kept as low as possible. Should the orebodies of a mining tract approach the stage of exhaustion, new mining ground containing ore must be acquired if the mining company is to continue its existence. To aid the operator in attaining the above important objectives is the chief function of the present-day geological department.

It may be remarked, however, that finding new mines has proved to be a much more difficult task in recent years than it was during the early days of prospecting in the West. The veins first discovered by the prospectors were generally marked by well-defined or easily traced outcrops. It was not uncommon to find profitable ore at or near the ground surface. In late years ore finding has become increasingly difficult, with a much greater financial risk to the miner and prospector. This condition has resulted in greatly increased demands, among those in search

of mines, for specially trained geologists. Many of the larger companies now maintain geological departments.

Among the first to recognize the need for geological study in connection with mine development and operation was the late Horace V. Winchell, who was engaged for geologic study and mapping in connection with one of the early apex controversies in the Butte district, and under whose direction the Geological Department of the Anaconda Copper Mining Company was formed in 1900. Winchell's efforts received the enthusiastic support of the late D. W. Brunton, then consulting engineer to the Amalgamated Copper Company and one of the first of the mine operators in Butte to realize the advantages to be gained in the use of mine geologic maps and models as practical aids to daily mine operations.

As first organized, the Geological Department consisted of three members, a chief geologist, one assistant geologist, and one draftsman. The remarkable complexity of Butte geological conditions was not, at the time, fully understood, because of limited underground development. Ore was plentiful everywhere; consequently to the mine foreman there appeared to be little need for geological advice.

The important Blue system of fault veins was first recognized as a series of faulting fissures by Winchell in 1900, although the district had been an important producer for more than fifteen years. In 1898, the Rarus fault had played an important part in apex litigation. Still later, the Steward and Middle faults were recognized, increasing the ore-hunting troubles of the miner. In every mine of the district one or more of these complex fault zones was found cutting and displacing the intricate and branching structures of the earlier formed veins comprising the Anaconda or East-West system.

As greater depths were reached and the miner's difficulties multiplied, the fact became evident that, without a more intensive geologic study, it would be impossible satisfactorily to solve the daily problems of the operator or deter-

mine the broader and extremely important questions of property rights. To meet these needs the Geological Department was enlarged to include all of the mines under control of the Amalgamated Copper Company. Later, as the operators began to recognize more fully the value and necessity of geologic mapping as an aid to mining, the mine manager and general superintendent required geologic maps of all mines for their personal use. Mine foremen used only composite working maps containing no geological information. At this time there was no control or directing of development work by geologists, whose duties consisted chiefly in mapping underground geology for the purpose of maintaining a complete geological record in the department and incidentally acting in an advisory capacity only in matters pertaining to mine development. During the period from 1900 to 1906 mapping in connection with litigation work formed an important part of the geologist's duties.

When, in 1910, the Anaconda Copper Mining Company absorbed many large producing companies of the district, it was soon apparent that the consolidation of the geological and engineering departments under one head would be a logical and satisfactory procedure. The next necessary step was to include the sampling department under the Geological-Engineering Department.

Thus the Geological Department of three members in 1900 grew to the combined Geological-Engineering Department of 1928, having under one control in the Butte mine operations fifteen geologists, forty-four engineers, sixty-seven samplers, and eleven draftsmen, exclusive of the heads of the various branches. The demands for geological service from the Mining Department had increased to such an extent that in 1928 one geologist was required for each operating mine, with an additional stope geologist in each of three of the largest producers, compared with one geologist doing the geology in four or five large mines in the early years of the Geological Department's existence.

In Butte, the relation of the Geological-Engineering Department to the Operating Department is chiefly one of service. The large amount of work assigned to the combined Geological-Engineering Department has made necessary a careful standardization of operations. At present sixteen operating mines, with an average total of 1,800 contracts, have to be measured weekly by engineers. In the larger mines over 3,000 samples are taken monthly. The mine geologist's work must be checked and new men given thorough training in geologic methods before being put in charge of responsible work. Better to meet these demands, the mines have been divided into two groups for geological and engineering supervision, and into three groups for control of sampling. Each of the engineering groups is directed by a chief assistant engineer; two experienced geologists are in direct charge of geological work; and a chief sampler and three assistants are responsible for the sampling. These men in direct charge report to, and are in daily conference with, the chief assistant geological engineer who has general direction of the work of the department in the Butte district. This plan of organization has resulted in standardized methods where

possible, and a co-ordination of the various branches of service, thus preventing duplication of effort.

The engineers make all necessary surface and underground surveys for the usual monthly posting of maps and stope books, and any special surveys required, such as crosscut, raise, or shaft connections. Engineers, assisted by samplers, measure all contracts weekly. From the underground surveys, stope books and complete sets of plan maps on 50, 100, and 200 scales are prepared for use in the geological and other departments where needed. Longitudinal vertical sections of all veins are posted semi-annually. The engineering office furnishes progress reports at regular periods, giving such data as working advances and cubic feet of excavation.

The geologist guides the miner to reach veins with the smallest possible amount of crosscutting; to keep drifts and raises in veins when it best serves the miner's purpose; and to keep shafts, drift laterals, and haulageways away from faults and bad ground. The geologist provides geological sketches, maps, and sections, which are made available to the foreman, shift bosses, and samplers at the mine for study; also, recommendations for work planned to develop known veins, discover faulted vein segments, and find new ore. In fact, the planning and control of all development work is assigned to the Geological Department. This is done through standard forms describing the work to be performed and the objectives sought. After approval by the head of the geological office, these sheets are sent to the general superintendent, to be issued later to the mine foremen. The geologist then watches the progress of the work until the objective is reached. A single crosscut project recommended by a geologist and begun in 1924 has resulted in the discovery of a series of orebodies worth many millions of dollars. Upon the geologist rests the responsibility of having development work laid out well in advance of actual construction, in order that the mine foreman may take advantage of such work to fit in with stoping operations, where stope filling is needed.

At Butte a most excellent co-operation exists between the geologist and those in direct charge at the mines. The geologist is at all times familiar with the plans of the foreman. His development recommendations are therefore made with due regard to the needs and requirements of the mine. That the mine foreman is appreciative of the value of the advice of the geologist is proved by his frequent calls for co-operative help.

For the manager and superintendent the geologist provides geological maps, estimates of ore, and the necessary development programs designed to prospect the company's undeveloped territory and to maintain ore reserves at the different mines.

Geologic maps prepared by the department consist of 50-, 100-, and 200-scale maps for office records; a complete set of separate level maps on 100 scale covering all of the mines of the district, for the general superintendent; a 100-scale set for each assistant superintendent covering the mines under his immediate direction, and 50- or 100-scale maps for mine foremen. To provide the most efficient service for the mining department, the geologists are required to post the mine foreman's maps in pencil, based on compass surveys, as soon as the geologic notes are taken, without waiting for the regular monthly transit survey. This pencil posting is erased when the final survey appears, but in the meantime the foreman has had the benefit of up-to-date information. The same service is extended to the general superintendent.

In addition, complete sets of 100-scale vertical geologic sections are maintained in the Geological Department, together with numerous plan and section maps dealing with special problems. Vertical longitudinal projections of all stopes are made and posted semi-annually, whether on veins, branch veins, streaks, or faulted segments of veins. Drift and raise assay averages are posted on these sections for ore calculation purposes. Mine models have also been found useful to illustrate geologic structure. Many types were developed in the early days of litigation to make clear to court or jury the respective claims of the litigant. Many of these are impracticable for daily use. The separate level glass sheet model has proved most satisfactory to the geologist, whereas the skeleton working model finds most favor with the mine superintendent and foremen. A well-equipped model shop is maintained in connection with the Geological Department.

One type of service furnished to the mine operating staff, including the mine foreman, assistant foreman, and shift bosses, is illustrated in the stope card system. These stope cards are of standard size, 10x6½ in. The stope

sampler's card shows a plan sketch of each floor of the stope on a 20 scale, with the number and position indicated where the sample was taken. A side column gives the sample numbers and assay results. A smaller attached card shows the assay-foot results from calculations made by the sampler of sampled sections which are spaced at ten-foot intervals. From these cards the metal content of the various bands and streaks of ore minerals and granite composing the stope face may be readily ascertained, as well as the average of the entire excavation. In addition to the sampler's cards, the foreman is furnished with a duplicate of the sampler's card of each stope floor upon which the stope geologist has carefully mapped the structural detail of the orebody showing veins, streaks, fault slips, and granite. The geologist's card is of material aid in keeping a proper orientation of the stope timbers and preventing ore losses caused either by branching of the vein or the presence of minor local faulting. A sample and geologic record of this character covering a stope floor is of great assistance to the foreman and shift bosses controlling ore-breaking operations, so as to derive a maximum profit from the ore block in process of stoping. As an aid to geologic study these records of stope geology are often of great assistance in deciphering complicated fault and vein conditions which may not be clearly disclosed in main level workings.

The Sampling Department is under immediate control of the Geological Department. At the mines the samplers are in charge of a senior sampler, who in turn reports to the office of the chief sampler. Drifts on veins are sampled every round and stopes every ten feet. Drift and crosscut sample results are recorded in ledgers, and assay curves on a 50 scale are made up for use by geologists. Sampling methods are standardized, but, because of the complicated internal vein structure and erratic distribution of ore minerals occurring within the orebodies, the results are naturally influenced to some degree by the personal judgment of the sampler.

Geologists are recruited as far as possible from the sampling and engineering departments. The general training and knowledge of geology and mining methods gained while working as a sampler are invaluable for the geologist fresh from college with but little underground experience. These men are then put on as stope geologists or as assistants to more experienced men and kept in these positions until sufficiently trained to be put in charge of the geologic work covering one or more mines. In all phases of the work, whether geological, engineering, or sampling, new men are generally required to have had an engineering education. Experience has shown that men trained in professions other than engineering do not get a satisfactory grasp of the problems involved in mine operations. At present there is but one man in the entire group of 133 geologists, engineers, and samplers who has not attended an engineering college.

Although great stress is laid upon the part played by the geologist in aiding actual mining operations, one of the main purposes of geologic study is to gain the broadest possible knowledge of the geology of the Butte district. The area known to contain fissures related to the fissure system of the productive territory measures eight miles in an easterly and westerly direction and five miles from south to north. Actually, a central area three miles by two miles has produced all of the copper and probably 90 per cent of the zinc output of the district.

Much study is given to all phases of the problems of ore genesis. A number of technical papers have been written by members of the Geological Department setting forth the results of these studies.

Surveys and maps made by Anaconda engineers in connection with timberland holdings and coal and oil lands in the state have been made available to the Bureau of Geology and Mines maintained in connection with the State School of Mines at Butte. This information will be incorporated with mapping done by the Bureau geologists into a geologic map of the state, to be published by the Bureau in the near future. A portion of the map covering the eastern half of the state has already appeared.

In conjunction with the geological work in the Butte mines, the examination of mineral lands, mines, and prospects forms an important part of the work of the Geological Department. The ever-expanding interests of the Anaconda Copper Mining Company have made necessary the examination of a wide range of ore deposits, including copper, zinc, lead, silver, gold, and nickel; also phosphate, coal, iron, manganese, oil, gas, bauxite, fluorite, and lime rock located in many parts of the world. A number of mines have been acquired, following such examinations. Among the more important acquisitions may be mentioned the North Lily, at Tintic, Utah; the Andes Copper property, in Chile, and the Giesche zinc-lead and coal mines, near Katowice, Poland. Geological departments, headed by men of Butte training and experience, have been formed at these newly acquired properties, thus affording to them the same service as given at Butte.

An important branch of the Anaconda Copper Mining Company's Geological Department is maintained at Salt Lake City in connection with the operations of the International Smelting Company and subsidiary mining companies. The chief object of this branch is to assist in maintaining a continuous flow of a variety of ores to the reduction plant at International, Utah, and to acquaint itself with mining operations in southern Idaho, eastern California, Nevada, western Colorado, and Utah, which are considered tributary to the plant at International. In addition to this, the duties involve the examination of many properties, either for purchase or for the purpose of checking ore reserves against loans or money advances made by the smelting company, or to assist the operators in development work where such assistance may be desired.

The International Smelting Company must be prepared to take small or large amounts of any class of ore offered. To maintain a cheap smelting rate, a balanced charge is necessary, and although ore receipts for one month may show a charge close to the desired balance, the month following may bring an excess of either siliceous or irony material. The Geological Department by a study of the various districts is called upon for information concerning the possibility of obtaining a supply of the type of ore needed.

# Evolution of Mining Practice at Butte

By WILLIAM B. DALY
*Manager of Mines*

BUTTE originated as a placer mining camp in 1864, when gold was discovered on Silver Bow Creek near the southerly end of what is now Main St. Placer mining was vigorous along this creek and its tributary gulches for about five years, after which, about 1870, the Butte camp was practically deserted. In the meantime, the quartz veins prominently traceable over the hillsides were located, the first location being made on July 12, 1864.

Reasoning from the presence of the gold placers, free-milling gold and silver ores were eagerly looked for by the early prospectors. Claims covering the great Rainbow and other silver lodes were located. The presence of valuable silver ores in the oxidized parts of these veins caused the staking of many claims, believed to be valuable only for silver, but which later, at depth, proved to be enormous copper producers.

Treatment of these early discovered rich silver ores was first attempted in arrastres, but later stamp mills were built. The first stamp mill was constructed in 1875, and others were built from time to time until a total of 290 stamps were working on silver ores. The silver mining industry reached its zenith in 1887, and declined rapidly in the early 90's, following the sudden drop in the price of silver in 1892.

During this period of active silver mining, the business of copper mining was making rapid strides. Development of many of the veins, located as silver veins, had uncovered copper ores. There was great activity in staking claims in the years from 1871 to 1879, when most of the claims covering the present productive copper area were located. The outcrops, or those portions of the veins nearest the surface, were oxidized or leached, and almost without exception were barren of copper. They were sought, therefore, for the silver they contained.

The Original, Parrot, and Mount Chief, located in 1864, were the first claims staked on what later proved to be valuable copper veins. The Anaconda claim was not located until 1875, when the location was made for silver ore. For a few years, a large tonnage of the oxidized part of this vein, being barren of copper, was treated in one of the large silver stamp mills. The early-day copper ore mined from the Original, Parrot, Colusa, Gambetta, and Mount Chief claims was hauled to the Union Pacific Railroad at Corrine, Utah, a distance of 400 miles, whence it was shipped to Eastern plants for reduction. Some chalcocite ore from the Anaconda claim was shipped to Swansea, in Wales.

The first successful copper smelter in Butte was the Colorado Smelting Company works, which treated custom ores and first operated in 1879. Following the building of this smelter, the development of the copper mines was rapid. Discovery of rich copper ores in the Anaconda and other claims led to the formation of a number of large operating companies, and soon four smelters—namely, the Parrot, Montana Copper, Clark's Colusa, and Bell—were built in rapid succession. The discovery of new producing mines quickly brought into the field the large smelting plants of the Butte & Boston, Butte Reduction Works, and Montana Ore Purchasing Company, and the removal and enlargement of the Montana Copper Company's plant to Great Falls in 1892, under the name of the Boston & Montana Company. In 1883, the Anaconda Mining Company began the construction of the Anaconda smelter near Anaconda, which was completed in 1884 and was superseded by the present works in 1902.

Oxidized silver-bearing ores near the surface were nearly always mined through inclined shafts sunk on the dip of the veins. These shafts were shallow, for they were sunk only to the sulphide ore, usually less than 100 ft. deep. Drifts were driven from the shaft on the vein and crude methods of stoping were employed. Drilling was done by hand and all of the ore was trammed by man power. The ore was hoisted to surface by windlasses with man power, or by whims operated by horses. All supplies were lowered in the shafts by the same methods. Mining engineers and geologists were unknown in the Butte district at this time, and no records of underground excavations were made.

With the development of large reserves of copper ores at greater depth, improvements in mining practices were rapidly made. Vertical shafts were sunk in the granite formation instead of the inclined shafts on the dip of the veins. Stations were cut at 100-ft. intervals in the shafts and crosscuts driven therefrom to the veins. Drifts on the veins and raises therefrom to the level above were driven, and the ore was stoped by the horizontal overhand method with timber and filling.

Early in this period, all drilling was done by hand. Later, with the development of the piston rock drill and the installation of air compressors, all drilling was done by this method. Small steam-driven hoisting engines were installed, to be later replaced by larger ones. All ore and waste were hoisted in small mine cars, and supplies were lowered on cages. The water was pumped by individual pumping plants at each shaft. All ore, waste, and supplies were trammed by men or animals. Natural ventilation alone was employed, the shaft having the highest collar being the upcast, while the air was drawn into the mine through the shaft with the lowest collar. Mining accounting was crude in those days and efficiency was based alone on a cost per ton of ore furnished about a month after the period of operation which it covered. If the ton cost was low, it was lucky, but if it was high a foreman was usually discharged, whether he was faithfully performing his duties or not. The mining engineer, or transit man, became a part of the organization in the early stages of this period, but his work was confined principally to making surveys and "hanging lines" for directions. Later, his duties included the making of level maps and the posting of all surveys thereon, as well as keeping the records of the excavations in stopes. The geologist and geology were still unknown so far as mining was concerned, and the superintendent or foreman directed the development work in a more or less haphazard way.

From the time that copper ore was first mined in the Butte district up to 1899, the foregoing practices were carried on by seven or eight large operating companies without much improvement and without standardization of any kind. In 1899, the Amalgamated Copper Mining Company was formed, and all but one of the large operating companies were merged therein. This was the first real step toward a more economic operation of the mines and a standardization of mining practices in the Butte district. Each operating company, however, had too many shafts, there being nearly thirty in operation, and the levels and drifts on the many veins were on different elevations from those in the adjoining properties. The cost of driving

new levels connecting all the mines on uniform level elevations was considerably more than the saving that could be made thereby. Since then about half of the working shafts have been eliminated and are now used for ventilation only, and the objective in the future is still further to minimize their number. The 2,800-ft. level and all levels below will be on uniform elevations in all mines.

Among the many improvements made in mining practices since the merger in 1899 are the following: Drain levels are on uniform elevations connecting all mines, so that the water flows by gravity to two central pumping plants, where it is pumped; changing the steam-driven hoists to compressed air hoists, with the compressors operated by electric power, which was done prior to the development of large electric hoists, and which are now being supplanted by the latest type of electric hoists, capable of hoisting from the 5,000-ft. level; mechanical ventilation; motor haulage; elimination of excess or surplus shafts; hoisting with large skips instead of with cars on cages; central machine and blacksmith shops and heating plants; application of geology to development work and mining; improvement of maps and mapping of underground excavations; sampling and estimation of reserves; sampling of stopes for more intelligent and economic operation; stope analyses; ton-for-ton development program; safety-first work, and underground fire-fighting, equipment and rescue stations; backfilling and rill stoping; contract system; unit operation plan of shafts for better efficiency; installation of research laboratory and of efficiency department for testing all kinds of supplies; standardization of practices wherever possible; selective mining by breaking minimum amount of waste and sorting out of the waste broken; and improved supervision and cumulative costing methods from day to day on a pound-cost basis. These improvements will now be more fully discussed.

*

## *Present Methods of Mining, Haulage, Hoisting, Ventilation, and Other Important Phases of Operation*

DEVELOPMENT — Practically all of the important ore deposits in the Butte district occur in fissure veins and in mineralized faults in a granite formation. In many parts of the district, the country rock, for considerable distances from the walls of the vein, has been altered and decomposed. This condition is especially true along the fractures and joint planes in which this alteration, together with the adjusting movement, has taken place and has weakened the structure so that the walls of the stopes, and even the drifts and crosscuts, lack self-supporting cohesiveness, causing crumbling in blocky masses. This tendency is greatly magnified by even small quantities of water percolating through the country rock or finding a way through the stopes from levels above. The continuity of rock support is practically destroyed in the planes of the numerous and complex post-mineral faults that are manifested in most occurrences by large gouge seams. Frequently these faults follow the walls of the veins, but more often they cut across them. These conditions cause much sloughing and squeezing of the walls and back, and demand close timbering and filling of stopes before long exposure to the air.

Shafts, drifts, and crosscuts, to be supported, must be driven of small size, which, in turn, prevents the use of large cars, locomotives, or mechanical loaders. The necessary surface structures and railroads regulate the stoping practice, on account of subsidence. These conditions also prevent the adoption of open stoping, shrinkage stoping, or any of the numerous caving methods, and restrict the size of shaft compartments and loading pockets that may be adopted without danger of prohibitive expense for jacket-setting and other maintenance.

The vein systems are complex and show much variation in strike and dip. Frequent displacements occur in many parts of the veins and orebodies for distances of from a few feet to 300 ft., by faults cutting the veins at all kinds of angles; thus the orebodies are divided into separate blocks of relatively small dimensions, greatly complicating development. Vein dips are, however, seldom flatter than 40 deg. from the horizontal, so that the ore, unless wet, will nearly always run by gravity on the slope of the footwall of the vein. Usually the walls of the veins are fairly distinct, though occasional irregular mineralization extends into the wall rock. On account of the complexity of the geological structure, unusually comprehensive surveys and geological mapping must be made, and, on account of the large amount of valueless sulphides in the copper ores, it is sometimes difficult to determine visually the relative values of the ores, so that careful and systematic sampling is necessary.

Development and mining operations are carried on through vertical shafts, which type was adopted instead of the incline shaft because of the steep angle of dip of the veins; the heavy ground in and near the veins, which makes the repair costs much less with the vertical than with the inclined shaft; and the fact that several decks of cages can be used in lowering timber and other supplies and in handling men, and cages can be easily and quickly changed to skips when ore is to be hoisted. The sizes of shafts vary somewhat, but the standard shaft is rectangular in shape and 20 ft. long by 7 ft. wide, outside dimensions. It is divided into two hoisting compartments, an auxiliary compartment and a compartment for pipes and cables. The two hoist compartments and the auxiliary compartment are each 5x4 ft. 6 in. and the pump compartment is 5x3 ft. inside measurement. In some shafts, a standard compartment is also carried for ventilation and a standard compartment for pipes and cables.

In selecting a permanent shaft location, the principal facts taken into consideration are: a rock formation free of faults and veins, wherever possible, and in ground showing minimum movement; close proximity to large orebodies; due consideration of the dip of the veins and faults; suitable location for construction of permanent engine, headframe, and ore-bin foundations; and the probable effect of underground operations with respect to the movement of the ground at the surface.

The general plan of development consists essentially in opening up levels at 100- or 200-ft. intervals (dependent on whether the veins are wide or narrow, and the distance from the shaft) and crosscutting the shortest distance from the shaft to the projected position of the vein or orebody that it is desired to open, and then to drift both ways on the vein. After the level development is completed, or simultaneously with it, raises are driven through from level to level, primarily for ventilation and to block out the ore, but later to serve as waste chutes for filling the stopes below, handling ore, and for lowering supplies.

Up to 1900, no consideration whatever was given to the geology of the district, or its application to development or mining. During that year the Anaconda Copper Mining Company established the first Geological Department. For several years thereafter the principal functions of this department were to study the geology of the district, take notes, make geological maps, and estimate ore reserves. This information was applied at a number of mines in the early stages of this period. The superintendents and foremen could direct the development work more intelligently with this information, and it proved so valuable that in 1914 the company adopted a standard practice which provided that no development work could be done at any of its mines without a recommendation from the Geological Department approved by the general superintendent of mines. The foremen and superintendents may make recommendations, but the work cannot be done until approved by this department. This practice has been in effect ever since.

The work of the Geological Department has been materially increased since

then, as is described elsewhere by Mr. Sales, and at present the department has full charge of all sampling done underground, whether it is in development work or stoping. It also issues weekly advance reports covering all development work, which show the footage advanced, the width of the ore, and the average assay value for the advance made.

To explore or develop extensive ore reserves in this district is impracticable, on account of the prohibitive expense in keeping up repairs. Each mine should, however, have developed ore reserves to enable it to operate at maximum capacity for several years, and it should do a sufficient amount of development work each year to keep these reserves intact. The necessary amount of development which should be done each year cannot be measured in dollars and cents or by the cost per ton of ore extracted. It must be an amount that will develop a new ton of ore for each ton extracted. For this reason, the Geological Department has initiated what is known as a "ton-for-ton program." Under this program, this department issues a sufficient number of recommendations to meet the requirements, and the results obtained therefrom are carefully checked and estimated as the work progresses. At the end of each quarter the results of this program are estimated, and if the tonnage of new ore developed is not equivalent to the tonnage of ore extracted, the amount of development work is increased proportionately. Under this program, the ore reserves at each mine are always kept at a figure that will enable the mine to maintain its normal or maximum production until all of its ore is exhausted.

DRAINING — With the exception of a few mines outside the central area, all mines of the Anaconda company are connected underground with drain levels, the water flowing by gravity to two central pumping plants. Practically all of the pumping in the district is done by electric plunger pumps of the vertical and horizontal, quintuplex and triplex, geared types. Because of the corrosive action of the water in some of the mines, many of these pumps are fitted with bronze water ends and special acid-resisting plungers. The pumps are operated against heads as high as 1,200 ft. At the two central pumping plants the water is raised from large tanks cut in the granite around and beneath the stations. The water columns are usually made up of 10-ft. lengths of pipe with flanges shrunk and peened on; they are lined with $\frac{3}{8}$-in. sheet lead or with $\frac{5}{8}$-in. or $\frac{3}{4}$-in. sectional wood lining for the corrosive water, but left unlined for the non-corrosive water. They range in size from 6- to 11-in. pipe. For the most part, standard pipe is used for lifts up to 1,000 ft. and extra strong pipe for greater lifts. An extra water column is also carried in the two main pumping shafts. The water contains a small percentage of copper in solution, which is recovered at low cost per pound in launders on the surface by precipitation. The recovery averages 95 per cent, and a considerable amount of copper is produced from this source each year.

STOPING — Levels throughout the district are generally spaced 200 ft. apart vertically and drifts are driven on the veins on each level, though if the orebody is wide and close to the shaft, 100-ft. intervals are used. The drifts kept open for haulage are usually 5x7 ft. in the clear. Stoping methods in general use are square set, timbered and untimbered rills, stull, and backfilling. The chutes in square-set stopes are placed 25 ft. apart and in the rills from 50 to 150 ft., depending on conditions. These stoping methods are more costly than the top-slice, shrinkage, or any of the caving methods, which have all been thoroughly studied, but, owing to the character of the veins, faults, and country rock in which the veins occur, cannot be used successfully in this district. The stopes are filled with waste obtained from sorting in the stopes, from development work, and from drifting on barren veins. This filling from development work and barren veins is dropped into the raise chutes from the level above, drawn off from a temporary chute mouth at the top of the stope, and permitted to run into place. The short block system of stopes is in general use.

The drifts are drilled with the latest types of drifters, the stopes with self-rotating and hand-rotated wet hammer drills, and the shafts with jackhammers. In drilling in drifts and crosscuts, the general practice is the use of the usual downcut round of from nine to fourteen holes, depending on the hardness of the ground. This round is drilled from one set-up of the bar. In the stopes, most of the holes are drilled from the same floor.

One-inch quarter octagon drill steel for both drifters and stopers and $\frac{7}{8}$-in. hexagon for jackhammers are used. For two years, the company expended considerable money in developing the Hawkesworth detachable bit and later the Hawkesworth Drill Company carried on further work in perfecting it. About four years ago, one mine was fully equipped with this bit, and two other mines were equipped later. Careful time studies and tests were made and costs were kept, which demonstrated conclusively that the bit was a mechanical success; that it was safer in handling in the shaft and in stopes and raises; that it eliminated entirely the loss of time of the hoisting engine and in the shaft caused by lowering and hoisting the old drill steel, and the time spent by the men in getting the old drill steel in and out of the raises and stopes and to and from the shaft; and that it was more economical. As a result, the company standardized on the Hawkesworth bit in all of its mines in this district, purchased the plant of the Hawkesworth Drill Company, and now manufactures the bits and shanks for its own use. At present 70 per cent of the drilling is done with these bits and eventually all drilling will be so done.

The grade of explosives in general use is 30 and 35 per cent gelatin in stopes, drifts, raises, and crosscuts, and 60 per cent gelatin in shaft sinking.

Of the stope timber used in Butte, the larger part is framed at a large central framing plant at Rocker, situated about three miles from the mines and on the tracks of four convenient railways. Practically all of the mines have their own timber-framing shops, which are used to frame miscellaneous timber, but all standard timber is framed at Rocker and then shipped by railroad to the mines. No timber is framed underground. When the timber is received at the mines, it is stored on skids in the mine yards.

Timber used in drifts is generally round peeled native fir and is of superior quality and grade to the stope timber. In drifting, the tunnel set is used. The posts are 8 ft. long and from 12 to 18 in. in diameter. The caps are 5 ft. 4 in. over all, have a diameter of from 12 to 18 in., and are stood on 5-ft. 4-in. centers. Stope timber is of pine, round, and from 8 to 12 in. in diameter. Posts, caps, and girts are framed on both ends, and the sets are stood with 5-ft. 4-in. centers. The sill floor is covered by sheeting, is carried about 30 in. above the top of the sill set, and is supported on 10x10-in. caps resting on pony sets or blocks. Some of the sill timber is treated with arsenic as a preservative. The shaft timber is of Oregon or Montana native fir and is also treated with arsenic.

Selective mining has been in general use in the company mines during the last five years. This term does not mean here, as it sometimes does, the preferential mining of the higher-grade ore reserves. It means the mining of the ore with a minimum breakage of waste and the sorting out from the ore of such waste as is broken. The ore produced from a mine should have the same grade as the average grade of the available reserves. If higher, it is evident that the higher-grade stopes are being favored, and if this practice continues for any length of time, the average grade of the remaining reserves will be decreased and in time the grade of the ore produced must also decrease. On the other hand, if the grade of the ore produced is lower than the average grade of the reserves, it is evident that the ore is being diluted with waste, or that the lower-grade stopes are being favored. In either case, the pound cost will fluctuate, will not represent the true pound cost which the mine should make, and, in my judgment, is not sound business. The Geological Department not only estimates the tonnage of reserves, but the average grade as well. In the operation of the Butte mines, the average grade of the ore produced is always kept as nearly as possible the same as the average grade of the ore reserves.

Many factors that affect the pound cost enter into the operation of a copper-mining company, but the cost per ton of production and reduction, and the grade of the ore produced, are the two important ones. Frequently, the grade

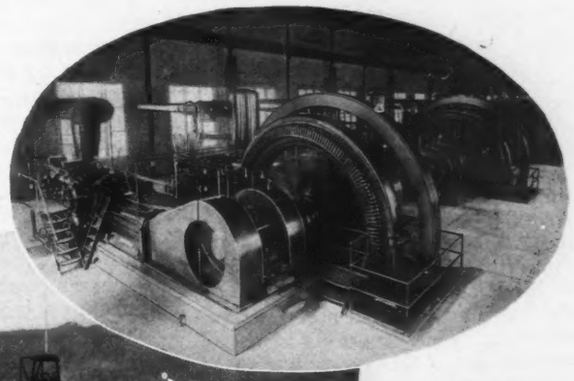
of the ore produced is much more important than the ton cost of production, but, unfortunately, too many mining companies in the past have given all of their attention to reducing the ton cost and little, if any, to the grade of the ore. The dilution of the ore with waste, or low-grade material that will not pay its way through, reduces the metal content of the ore produced much more, in proportion, than the reduction in the ton cost which is obtained by the larger tonnage of ore and waste together, and an increase in the pound cost is the inevitable result. The copper obtained from the waste, or other low-grade material, that will not pay its way through from the place where broken to finished copper, is produced at a loss, and this loss, although not apparent on the surface, decreases the profit on the commercial ore that is diluted by it. And again, this same low-grade material takes the place, throughout extraction and reduction, of ore that could be produced at a profit, and, as a result, the possible profit of the mine is further reduced. When pure waste is shipped with the ore, the output of the mine is reduced by as many tons of commercial ore as the tons of waste hoisted in the ore.

For this reason, in the mines of the company at Butte, the minimum of waste is broken, and such as is unavoidably present is sorted out and left in the stopes for filling, where it belongs. The stopes are sampled every ten feet longitudinally and in cross-section, so that the grade of the ore is definitely known by every one in the organization down to and including the shift boss.

A set of tables based on varying ton costs, on varying grades and gross values, and on varying selling prices of copper is provided for each mine. When the assay value from the sample of a given stope is determined, a simple calculation will indicate whether the ore will pay to stope or not. By this method, each stope is analyzed separately and must stand or fall by itself. If a mine cannot obtain its normal production from ores of sufficient grade to show a profit after the total costs of production and reduction, including fixed charges, are debited against it, then it is profitable to mine lower-grade ores, up to the capacity of the plant, against which only the cost of stoping, and in some instances tramming, are charged. By this reasoning, the Butte mines are enabled to output a considerable tonnage of very low-grade ore at a profit with the present price of copper, and that is being done.

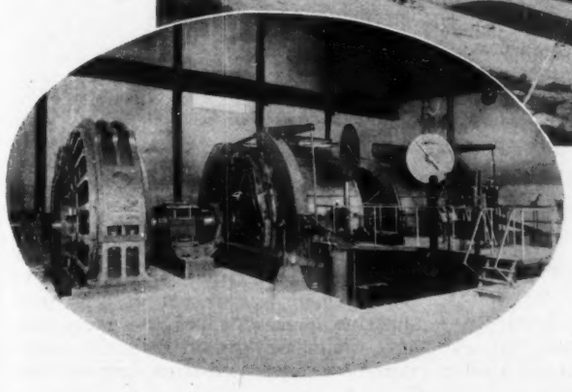
For a great many years, man and animal tramming prevailed in the Butte district. Later men and animals were replaced by the trolley locomotive, and now the mines are standardized with the storage-battery locomotive. The locomotives weigh 11,000 lb., have a drawbar pull on a level track of 1,500 lb. at a speed of 3½ miles per hour, and a maximum drawbar pull, for starting and accelerating, of 2,500 lb. The general over-all dimensions are: 36 in. wide, 51 in. high, and approximately 12 ft. long over the bumpers. They are charged between shifts in charging panels located at the stations.

VENTILATION—Without movement of air underground, life there could not exist. This movement must also include the pulling out of the impure air carrying carbon monoxide gas, powder smoke, other impurities and moisture, and the drawing in of pure dry air from the surface to replace it. Impure air carrying carbon monoxide gas in quantities above 0.0004 of 1 per cent, or pure air carrying a high percentage of humidity, is enervating to animal life, and an underground workman cannot perform a reasonable day's work under these conditions. The movement of air in most underground



Modern compressor plant at Butte

Surface plant at Badger State mine, Butte, with main continental divide of the Rocky Mountains in the background



Electric hoist at the Belmont mine, capable of hoisting from a vertical depth of one mile

mines is sufficient to remove the carbon monoxide gas, smoke, and other impurities, but, with rock temperatures increasing with depth, the question of humidity is the all-important one.

A man may perform a good day's work underground, as on surface, with temperatures ranging from 90 to 100 deg. F. if the humidity is 75 per cent or under. But, if we should reverse these conditions, and have the temperature at 75 deg. and the humidity between 90 and 100 per cent, or complete saturation, the man would become fatigued in a very short time. These conditions are noticeable in some of the large Eastern cities in summer time. It is a healthy condition for a human being to perspire freely under high temperature, and he will not be fatigued to any great extent by it if the air is dry enough to permit immediate evaporation.

The mines in the Butte district operated fairly successfully with natural ventilation for a number of years, although every mine had a large number of so-called "hot-boxes." The shafts with the highest collars, whether they were operating shafts or not, were the upcasts, or out-takes, and the shafts with the lower collars were the downcasts, or intakes. Under this method many operating shafts were upcast, which is not desirable.

As depth increased, rock temperatures were higher, the increased temperature promoting more rapid evaporation of water, with resultant increased humidity. During 1915, this condition became quite serious. A few suction fans had been installed prior to that time, but no general plan for relieving the condition had been formulated. During that year, an intensive survey was made. At that time, approximately 1,500,000 cu.ft. of free air per minute was taken into and out of the mines. The survey determined that this quantity must be doubled to bring about a satisfactory condition. At first it was thought necessary to sink fifteen or twenty more air shafts. The expenditure of this money was saved by smooth-surfacing the existing air shafts with concrete slabs, which reduced the coefficient of resistance by one-half, increased the velocity of the air, and doubled the volume without increasing the pressure or the power needed for the operation of the fans. At the same time it made the shafts fireproof.

During 1916 and 1917, all of the air shafts belonging to the company in this district were equipped with large reversible suction fans driven by electric motors, the air shafts were smooth-surfaced, the quantity of air taken in and out of the mines was increased to over 3,000,000 cu.ft. of free air per minute, and very satisfactory ventilating conditions were obtained. The reversible feature of the fans insures positive control of the air movements underground in case of fire. With this positive control, all operating shafts are made downcast and the air shafts are upcast. Our objective at all times is to keep the temperature at or below 80 deg. F. and the humidity at 75 per cent, or less, in all working places. The ventilation problem has been satisfactorily solved in the Butte district, and it will be possible to go as deep as the ore may be found, and still have satisfactory ventilating conditions.

HOISTING—The improvement in hoisting machinery has been marked. The company has standardized on one double-drum electric main hoist and one single-drum electric auxiliary hoist for each of the main operating shafts, which will hoist all of the ore in future. Three of these shafts are fully equipped with these hoists at present, and the others will be equipped from time to time. These hoists are each capable of hoisting 1,500 tons of ore during ten hours, from a vertical depth of one mile. The power for the double-drum hoists is supplied by a 2,150-hp. direct-current motor, directly connected, and the auxiliary hoists are serviced by a 1,000-hp. motor of the same type. The safety devices consist of Ward Leonard and Lilly controls. Owing to an abundance of power and the large load carried by the company, the flywheel has been eliminated in the construction of these hoists.

CENTRALIZATION OF SHOPS—Although each mine is equipped with a small blacksmith shop and machine shop for emergency work, the major part of this type of work, as well as boiler making, is performed at two central shops. One shop repairs all the rock drills. Two main heating plants supply nearly all of the heat for the surface plants.

CONTRACT SYSTEM—All of the different bonus systems in force throughout the country have been carefully studied, but the Anaconda company in Butte prefers its contract system, put into effect shortly after the World War. The system is based on a cubic foot of excavation and is made verbally for one week only, as, owing to the ever-changing conditions in each working place, it is inadvisable to let a contract for longer. All excavation work is done on contract and the contract price is fixed by the foreman at the beginning of each week. All contracts are measured and paid for weekly in cash or currency, and all men in a contract participate equally. All contracts, prices, earnings, measurements, and conditions are reviewed at a meeting held every two weeks by the general superintendent of mines, at which all superintendents and foremen are present. With this review it is impossible for a foreman to show favoritism to any contractor, or to show prejudice against him. Under this system, a man is paid for the work he performs. A good workman who is willing to work hard will earn good wages; the lazy workman will not.

UNIT PLAN OF SHAFT OPERATION—The shafts of the company in Butte are operated on the unit plan. All operating shafts are operated at normal, and sometimes maximum, capacity, and the non-operating shafts are maintained with the minimum amount of expense.

UNDERGROUND FIRE FIGHTING—An underground fire, particularly in a mine where much timber is used, is the most serious thing that can happen in mine operation, not only on account of the damage it may cause and the expense it incurs, but more particularly on account of the danger to employees and possible loss of life. In the early days an underground fire at Butte usually meant the loss of several lives, the closing of the mine for a long period, and sometimes the flooding of the mine with water. In later years, breathing apparatus, or oxygen helmets as they are sometimes called, were developed. Anaconda was probably the first of the metal-mining companies in the West to train men in the use of this improvement, and the U. S. Bureau of Mines, with its rescue car, has rendered valuable assistance, so that at present the Butte district boasts over 500 men thoroughly trained and experienced in this work. With trained men and a sufficient supply of breathing apparatus, an underground fire in its incipient stage may be fought just as firemen would fight a fire in a building on the surface, and is usually quickly extinguished. If, however, the fire originates in the old gobs from spontaneous combustion, or if the fire follows into the old gobs by burning the timber leading thereto, in either of which condition the fire is not exposed to view, a more serious situation exists. In this event the area is bulkheaded off, without interfering with the operation of the rest of the mine, and usually the fire is extinguished by pouring water through diamond-drill holes, drilled for the purpose, into the area affected. The airtight bulkheading of the area prevents fresh air from entering the fire zone, and allows carbon monoxide and carbon dioxide to develop in large quantities, thus extinguishing any open or exposed fire. In one instance, the methods described failed and the fire was extinguished by diamond drilling the entire area that was burning and then flooding through these drill holes large quantities of water carrying about 23 per cent of concentrator tailing. Two rescue stations, fully equipped with breathing apparatus, are manned day and night and are always ready for emergency calls.

LABORATORY AND EFFICIENCY DEPARTMENT—The company has established a research laboratory and efficiency department whose function it is to cooperate with the operating and purchasing departments and to examine and test all kinds of supplies as to grade and quality. It also salvages and reconditions supplies and parts for further use, and standardizes supplies for all mines.

IMPROVED SUPERVISION—In the earlier days, the personnel of the organization at each mine or shaft consisted of a foreman and shift bosses. Later, assistant foremen were added. For many years, the only supervision above the foremen was vested in a general superintendent of mines and one assistant. Commencing with 1914, the supervision was increased from time to time,

until at present it consists of a manager of mines, one general superintendent of mines, an assistant general superintendent for every two operating shafts, and a foreman for each shaft, with as many assistant foremen and shift bosses as are required to give detailed supervision to all work performed underground. With this increased supervision, each working place and the results therefrom are checked, the development work is more intelligently and efficiently prosecuted, the ton costs in the stopes are brought down to the minimum, the grade of the ore therefrom is maintained, each stope is carefully analyzed from the standpoint of whether it will pay or not, and a considerable amount of money above the cost of this increased supervision is saved each year.

ACCOUNTING AND COSTS—The operation of a large mining company is just as much a business as is the operation of a railroad, a manufacturing plant, or any other industry, and it should be conducted along sound business principles. It is not sufficient to make a mine pay a profit: it should be made to pay all that it can legitimately. If copper is sold by the pound, it should be produced on a pound-cost basis.

As a business proposition, mining companies have established accounting departments whose function in part is to distribute the detailed cost of labor, supplies, sundries, and other charges on a ton-cost basis. With most mining companies, these statements and costs are not available for the management until late in the month following the period which they cover. They merely record and show what happened. The opportunity to correct conditions which may have caused a high ton cost has passed and the only thing left to be done is to try to do better next month. The pound cost is not known by the management of the operating department, in most instances, if at all, until some time after the ton cost.

The operating manager should know from day to day during the current operating month approximately what his ton cost is, what the average grade of the ore is, and last, but by no means least, what the pound cost is. With this information, if the pound cost is higher than it should be legitimately, the management can correct the condition which causes it when it is happening and bring it back to where it should be. Without this information, the operating management may not know of the higher pound cost until it is too late to correct it, which is no way to operate a mine intelligently and economically. For this reason the Anaconda Copper Mining Company at Butte has adopted a cumulative costing method which shows from day to day during the current operating month what the ton cost is, and what the average grade of the ore and the pound cost is for each mine and for all mines. The degree of accuracy of these estimates when they are compared with the finished and completed figures furnished later by the accounting department, is surprising. Ton cost will not vary more than a few cents per ton and the pound cost will vary considerably less than a tenth of a cent per pound so far as the mining operations are concerned. The cumulative ton-cost estimates are based on the same accounts as appear later in the completed cost statements. In addition to these estimated costs, a cumulative ton cost is carried for each working place in the mine, so that the average ton cost is known. With this information, the superintendent or foreman may know if the proper efficiency is being obtained, and the information enables them to make comparisons with other comparable places.

# Timber and Lumber Interests

By W. C. LUBRECHT

*Manager, Lumber Department*

THE ORIGINAL timber resources of the Anaconda Copper Mining Company were acquired in the Bitter Root valley prior to 1896, but, with the future of the Butte mining district in view, a short time later the company purchased from the Northern Pacific Railway some of the timber lands in western Montana that had been secured by that company under United States land grants. This purchase covered the odd-numbered sections only, and, following this acquisition, much of the adjoining acreage has been added through purchase from others. Some extensive land exchanges have also been made, all of which have resulted in more consolidated holdings. These consolidations will naturally bring about improved operating conditions in the future, and block out the timber holdings to a point where no concern need be felt relative to an assured reserve of forest products for the mining district for an indefinite period.

Timber holdings of the Anaconda company are located in that section of Montana included between the continental divide and the Idaho state line, all being tributary to the Clark's Fork of the Columbia and Kootenai rivers. These timber stands are the closest merchantable timber bodies with relation to the Butte district, and are being administered with a view of maintaining a logical supply for the future requirements of the mining and milling operations that are tributary to Butte and Anaconda.

These forests predominate in Pondosa pine, Douglas fir, and Western larch; with spruce, lodgepole pine, and hemlock appearing in important quantities in some of the districts. The largest acreage of the company holdings is in Missoula County, the relative importance of the other counties being Lincoln, Sanders, Powell, Flathead, Mineral, Ravalli, Granite, Lewis and Clark, and Lake.

The total area in the Anaconda holdings is 954,554 acres, of which 228,515 acres are cut-over or partly cut-over lands, indicating about 726,039 acres of virgin timber lands. Merchantable timber available from these lands is estimated at 4,940,000,000 ft., board measure.

LUMBER PLANTS — During the last thirty years Anaconda has conducted widely distributed lumbering operations in western Montana. In the earlier years much of the lumber required in the Butte district was manufactured by the smaller units or portable mills, but this method resulted in a large loss in the conversion of the raw material from the forests and was naturally succeeded by investment in larger plants more centrally located. Modern plants, in their day, were built or acquired at Hamilton and St. Regis, Mont., and at Hope, Idaho.

The log supply for these plants generally was carried to the mills by log-drives through the natural tributary streams. This method, in turn, was found wasteful on account of the necessity of storing logs for long periods at the mills, resulting in deterioration in quality and the positive loss in the value of the lumber available for the general markets.

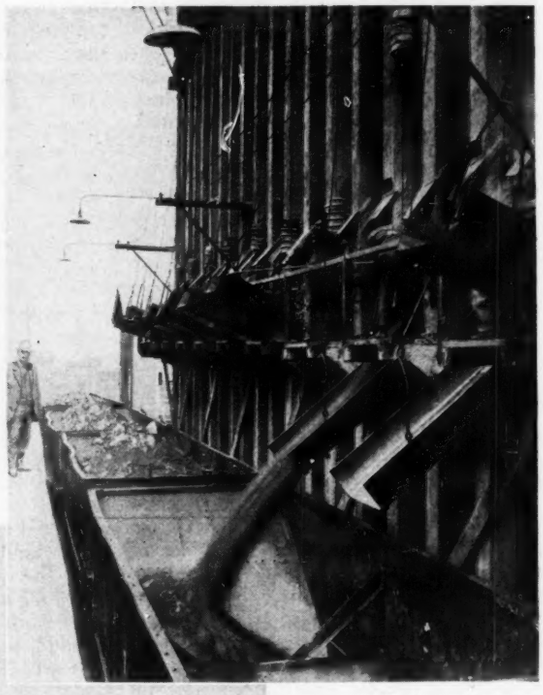
Following this came the trend toward transportation of the log supply by rail from the forests to the mills. This increased the unit cost of raw material somewhat, but resulted in reduced investment in log stocks and a higher quality of lumber from the logs. The concentration of manufacture at the Bonner plant followed the hauling of logs by rail, as better results were assured from one well-developed plant, operating steadily, than could be expected from three or four plants operated as conditions seemed to justify.

The Bonner mill contains three double-cutting band-mills as its main equipment and is modern in every respect. It has a capacity or actual production record of 210,000 b.ft. per eight-hour shift. With the addition of a part-time night shift, the annual cut has averaged 90,000,000 b.ft. under normal conditions. Yarding, planing mill, and remanufacturing facilities adequate for the sawmill capacity are provided at the Bonner plant.

The Milltown plant, acquired in 1928, with a main equipment of two double cutting band-mills, has a one-shift capacity of about 130,000 b.ft. and an average annual production of 35,000,000 b.ft. This plant is well provided with facilities necessary properly to prepare the output for the general markets. The Bonner and Milltown plants are so close together that the possibility of combination is promising.

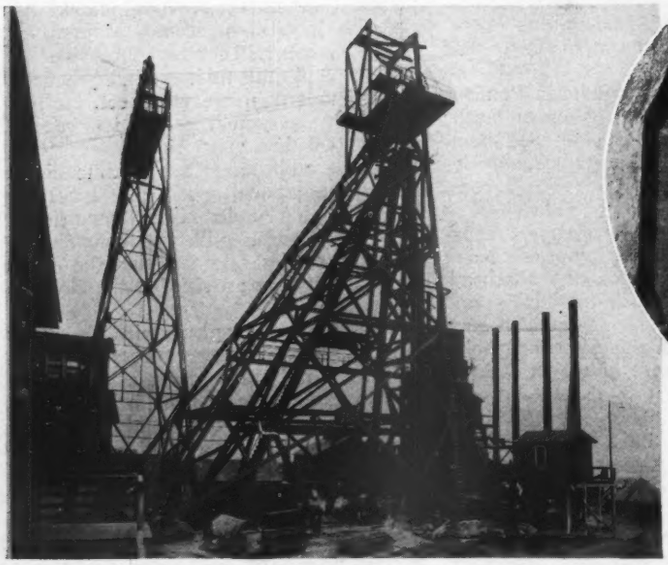
One section of Butte hill

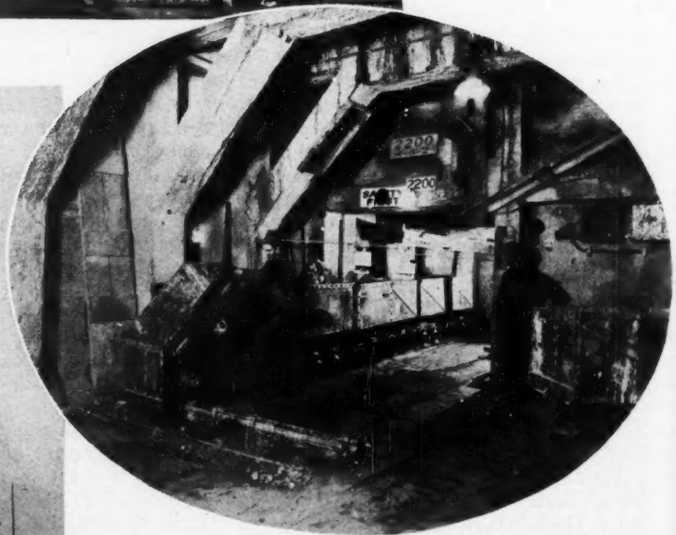
Loading ore from bins into railroad cars for shipment to Anaconda concentrator



Headframe and timber at the West Colusa mine

The new Belmont headframe, at Butte






Dumping station underground, in Butte

# Modern Metallurgical Miracles

*Progress from steam-stamp concentrators to ball mills and selective flotation, and from tiny matte furnaces to huge reverberatories—Making ore out of valueless material by pioneering in hydrometallurgy*

By Frederick Laist

ANACONDA'S developments in copper metallurgy almost constitute a history of that art in the United States during the last fifty years. From the time the first ore was hoisted in Butte until now, there is an unbroken record of achievement and of progress. The earliest reduction plants took their smelting practice and equipment from Swansea and their concentrating from Michigan, but in a remarkably short time the units were so greatly enlarged and the practice underwent such radical changes that only slight resemblance to the originals remained.

From the steam-stamp concentrators and the tiny matte furnaces of the early days of Butte and Anaconda to selective flotation and the great reverberatories and converters of today is a long step. The path of progress is littered with carefully thought out processes and devices which have gone into the discard after serving their purpose as stepping stones to greater and ever greater perfection. Anaconda's metallurgical history reminds one of that of iron and steel in its never-ending change and in the willingness of its captains to throw out equipment costing millions to make room for something better.

The illustration on p. 288, elaborated from a paper by E. P. Mathewson on reverberatory smelting, well illustrates the development of matte smelting at Anaconda and shows the evolution of the reverberatory from the pygmy of the early 'eighties to the giant of today. Nor is the change merely one of size; as the furnaces grew, the mode of operation was improved, so that out of the early wasteful intermittent types came the modern efficient continuous furnaces, with their steam boilers for recovery of heat from gases which formerly flamed luridly from the tops of smelter stacks, and their accessory mills for powdering coal for burning instead of allowing it to drop in part through grate bars and to waste.

For the first ten years or so, the Anaconda metallurgists were content with producing a high-grade matte, which they shipped to England for conversion into copper. Than the Manhes converter appeared on the horizon, and they were quick to see its possibilities and to undertake its development. In the hands of Anaconda men it grew in size and improved in mechanical perfection until the final touch, the basic lining, was added by a rival but a friendly group. The freemasonry which exists between metallurgists in our country is one of the potent causes of our pre-eminence in this field. However keen may be the business competition between companies, the interchange of information between their metallurgists is generally unhampered.

Along with changes in smelting practice, the preliminary treatment of the ore gradually deviated more and more from early models. The steam stamps with which the first mills were equipped, although admirably adapted to the crushing of ore containing native copper, were found to slime the brittle sulphide minerals of the Butte ores, and for many years the slime problem was the *bête noir*. Crushers and rolls arranged for comminution of the ore in stages, with concentration in between, were found to make less slime and thus increased the recovery. Classification and screening were improved so that each set of jigs and tables received a range of sizes best suited to its special characteristics. Gradually, recovery from coarser sizes and from sand improved, until the mills were making tailings which approached the furnace slags in copper tenor, but still the slime problem remained.

This was the situation when the new Anaconda mill was built at the Washoe Reduction Works in 1902, and so it remained for ten years more. The huge slime ponds below the works are mute but eloquent evidence of the importance of the problem. Slime leaving the mill was twice as rich as the porphyry ores of today and contained as much copper as jig and table tailings combined. The ponds were filled and excavated and filled again, with intervals of wastage in between, and still no solution had been found. Numerous devices were suggested and tried—always in vain. Extensive and costly experiments were made with centrifugal machines, without success. Finally, a partial solution was found by constructing an exceedingly compact form of buddle with decks of special composition and construction. This raised the mill recovery from 78 to 83 per cent.

About that time the flotation process began to attract notice, and experiments were started. It soon became apparent that this process would be revolutionary in its results, and in 1914 the application of the process to the Anaconda mills was undertaken. Many of the jigs and tables were retained, because the grade of concentrates remained the same, and since the ratio of concentration was only 3 into 1, it was desirable to produce as much coarse concentrate as possible for the smelter. This remodeling effectually solved the slime problem and resulted in raising the mill recovery from 83 to 96 per cent. At the same time, it had a far-reaching effect on smelting practice, since the high recovery made possible by flotation concentration did away with the necessity of sorting the ore at the mines into first and second class, and thus sounded the death knell of the blast furnace.

During the years 1904 to 1914, reverberatory practice at Anaconda had been steadily improved. At the time of the abandonment of the "Old Works," about 1902, these furnaces had already grown to respectable size and were operated in a more or less continuous manner, but at the "New Works"—that is, the Washoe Plant—development was even faster, and the furnaces were greatly elongated beyond anything that had been conceived of before. They grew from a length of 50 ft. to 116 ft., with but little

increase in width. The benefits derived from this change were astounding. The new furnaces made cleaner slags, required less fuel and labor per ton of ore, and smelted vastly greater tonnages. From a necessary evil, to be considered only for the smelting of material too fine to be charged into a blast furnace, the reverberatory became a powerful rival of the former.

The pre-eminence of the reverberatory became assured when coal-dust firing was successfully applied to the long furnaces in place of the wasteful and laborious method of firing by means of coarse coal burned on bar grates. Substitution of coal dust for coarse coal so greatly added to the economies which the lengthening of the furnaces had brought about that the unit cost of smelting in the reverberatory was fully as low as in the blast furnace, and this, when taken together with the extremely high mill recovery made possible by flotation, made it cheaper to concentrate all ore and thus doomed the blast furnace to extinction. The blast furnaces struggled along till shortly after the armistice was signed in 1918, kept alive by the need for production which was paramount during the war. About January of 1919, they were shut down, never to start again, and another plant, which only fifteen years before had cost millions to build, was written off to "obsolescence"—and to progress.

With the mill recovery at 96 per cent and an over-all recovery from ore to anodes of 93 per cent, it was obvious that little further could be done in the direction of improved recoveries. For long, however, the metallurgists at Anaconda felt annoyed at having to content themselves with producing, from an ore containing approximately 4 per cent of copper, a concentrate with only 10 per cent. Many attempts were made to drop the pyrite, to the presence of which the unsatisfactory grade of the concentrate was due, but although some of these attempts were successful metallurgically, they were not so economically, and the copper losses which accompanied the early attempts to raise grade invariably overbalanced the corresponding smelter savings.

The earliest flotation reagent used at Anaconda was an acid sludge supplied by the Union Oil Company, of California. This was necessarily applied in acid circuit. It was followed by sodium xanthate and pine oil, also in acid circuit. Finally, an alkaline circuit using xanthate was tried, and from this as a starting point, selective flotation was gradually developed and improved, so that now the concentrates contain 25 per cent of copper. Although the tailing has gone up to some extent—namely, from 0.17 to 0.24 per cent copper—this is offset in part by smaller slag loss, attained by the production of less slag. The net result has been a drop in overall recovery of about 1 per cent, with only about half as many tons of concentrates to roast and smelt per ton of copper finally produced.

For many years a serious matter had been damage by smoke to the surrounding country. As long ago as 1903, a huge system of flues and dust chambers was constructed, but although these effected an improvement, they were no solution. When the Cottrell process was invented, Anaconda was among the first to investigate its possibilities. These investigations culminated in the building of the largest Cottrell treater installation ever constructed, either before or since. The design of the plant and treater was along novel lines which permitted a maximum gas treatment in a minimum of space and cost of plant. The plant handles as high as 2,500,000 cu.ft. of gas per minute with satisfactory clearance. It has effectually solved the smoke problem.

At the same time, work was undertaken with a view to utilizing some of the sulphur dioxide gases for the production of sulphuric acid to be used for treating phosphate rock, which occurs in abundance in the neighboring State of Idaho. This required the development of a process for making high-grade superphosphate which could be shipped into the Eastern states, as there was not a sufficient market for superphosphate in the West. This was successfully accomplished, and Anaconda Treble Superphosphate has now been on the market for a number of years. Ammonium phosphate is also being made.

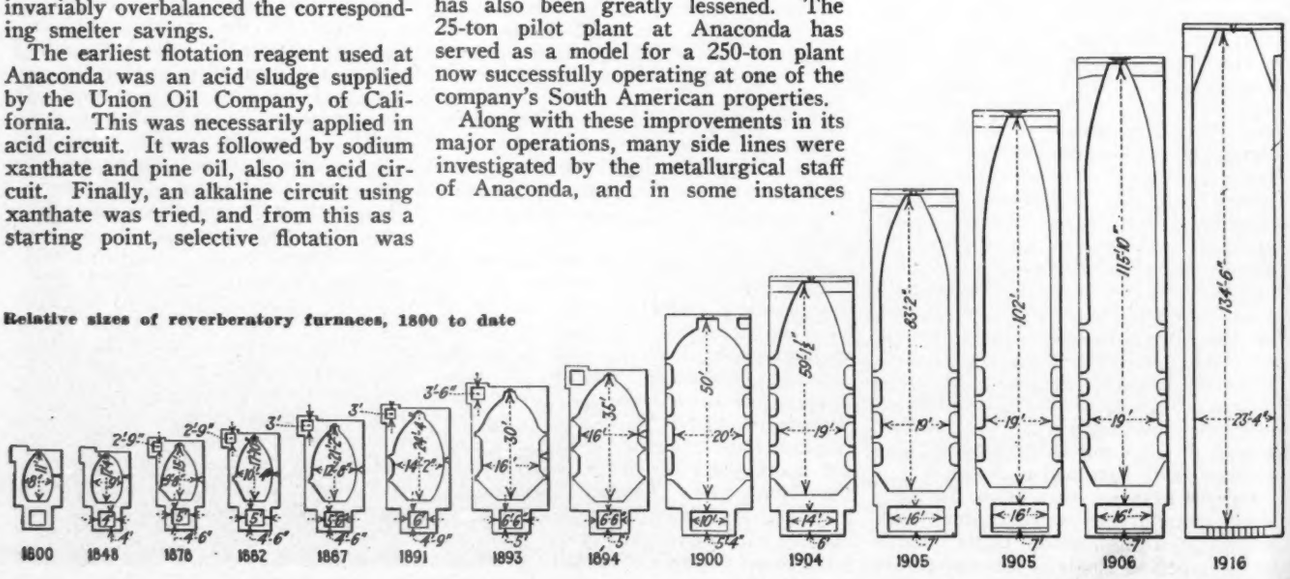
In connection with the phosphate process, a new type of acid plant was designed and built. Towers are used instead of chambers, and as a consequence the area required has been reduced enormously and the first cost of plant has also been greatly lessened. The 25-ton pilot plant at Anaconda has served as a model for a 250-ton plant now successfully operating at one of the company's South American properties.

Along with these improvements in its major operations, many side lines were investigated by the metallurgical staff of Anaconda, and in some instances valuable results were achieved. One of these was the development of a plant for the treatment by leaching of low-grade cupriferous material. As a result of water concentration, many millions of tons of tailing containing about 13 lb. of copper per ton had accumulated. This material was partly oxidized and hence not well adapted to re-treatment by flotation, even had this process been available in 1910, when the project of beneficiating this material was first conceived.

Although, at that time, leaching was not viewed with much favor by copper metallurgists, it was decided to attempt its application to this tailing. The problem seemed and was formidable, for the copper, being present chiefly in the form of sulphide, had first to be made soluble in acid by roasting. The process of roasting, leaching with dilute acid, and precipitating the copper from solution by scrap iron, was simple enough chemically and presented no novelty, but to apply it economically to such low-grade "ore" was not so easy. To make a profit sufficient to amortize the necessary plant in reasonable time, it was required that the combined cost of reclaiming, roasting, leaching, and precipitating, including all expense of labor, fuel, and acid, must not exceed $1 per ton.

The roasting of such material was considered, in those days, to be an expensive operation, and examples from practice tended to support this view. However, it was shown that the fuel consumption could be kept around 3.5 per cent and that the furnace capacity was sufficiently large to keep the plant cost within reasonable limits. The manufacture of sulphuric acid at copper-smelting plants was a somewhat novel departure, although this was being done successfully at Tennessee. The leaching of large tonnages of cuprous material had not been done, and costs were consequently theoretical. Precipitation of copper by iron was practised, and pre-

**Relative sizes of reverberatory furnaces, 1800 to date**



sented no difficulties. Electrolytic precipitation was considered, but was ruled out on account of the foulness of the solutions and the presence of salt, which was used for dissolving silver. Other means of precipitation were also considered, but were ruled out, as it was considered unwise to complicate an operation which, although simple chemically, contained so many unknown economic elements. This has been a cardinal principle at Anaconda and is one reason why there have been but few failures. To progress cautiously is to progress surely. To pyramid experiments is as dangerous as to pyramid stocks.

The copper-leaching plant at Anaconda was the first commercially successful plant of the kind, and operated on a lower-grade feed than was ever treated before or since. It was a distinct departure from the Henderson plants for treating Spanish pyrite cinders, and in magnitude of operation, in general procedure, in character of equipment, and in cost of operation it may be considered as having paved the way for the treatment of low-grade oxide copper ores which for so many years had been held worthless.

Another development of considerable importance and far-reaching effect—this time on the zinc and indirectly on the brass business—was the electrolytic zinc process. Nowhere is the saying "There is nothing new under the sun" more true than in the art of metallurgy. As a rule, the broad chemical principles of metallurgical processes are simple, and consequently have been known from time immemorial. So it was with electrolytic zinc. I do not know who first thought of leaching roasted zinc concentrates with dilute sulphuric acid and recovering the zinc as metal from solution by electrolysis, but the first to make a serious attempt to do it commercially was Letrange, about 1883. He failed for various reasons which it would be beyond the scope of this article to detail or discuss. Numerous others followed him, and then about 1898 came Ashcroft, who built a plant in Australia for the treatment of Broken Hill ores. He realized the importance of having pure solutions and otherwise seemed to have an excellent grasp of the problem, but he came too soon. His plant failed as a commercial venture, although he succeeded in making considerable zinc. The account of his work reads much like the daily logs of the experimental plant at Anaconda, with their record of good and bad days, of periods of optimism and depression. He was unfortunate in not having at his command the flotation process for preliminary preparation of his ore, the numerous devices for settling and filtering ore pulps which were invented in such numbers and variety in the years from 1900 to 1915, and, most important of all, cheap and abundant electric power.

The unprofitable outcome of Ashcroft's work discouraged others from further attempts to make electrolytic zinc from sulphate solutions, and for long it seemed as though the smelting of zinc concentrates in retorts would maintain its undisputed supremacy. Much work was done with dry chlorination processes, which enjoyed quite a vogue for ten or twelve years prior to 1912. They were investigated at Anaconda before it was decided to develop the sulphate process. Anaconda was not alone in its interest in electrolytic zinc, and about the time that experiments were undertaken here, similar work was being done at Trail, and at Bully Hill. There was some interchange of information, but for the most part the investigations were independent of each other.

### Co-operation

*"THE freemasonry which exists between metallurgists in our country is one of the potent causes of our pre-eminence. . . . However keen may be the business competition between companies, the interchange of information between their metallurgists is generally unhampered."*

Anaconda's progress was most rapid, stimulated as it was by the extraordinarily high prices which high-grade zinc brought in the early years of the war. Within a year after the preliminary laboratory tests, ground was broken at Great Falls for a plant to produce 100 tons of zinc per day. Since the construction of the Great Falls plant, the production of electrolytic zinc has steadily increased, until now it constitutes 21 per cent of the world's total zinc output and is still increasing. Besides the two plants of the Anaconda company at Great Falls and Anaconda, which have a combined output of approximately 1,000,000 lb. of zinc per day, there is the huge plant of the Consolidated Mining & Smelting Company at Trail (the result of the Trail experiments), and there are plants in Idaho, Australia, Poland, Italy, and Germany, and plants are building in Illinois, Canada, and Norway.

One of the incentives for the development of the electrolytic zinc process was the existence of large quantities of complex ores containing lead, zinc, silver, and gold. Such ores occurred in Montana, Idaho, and Utah in sufficient abundance to justify making a serious effort to treat them. In many instances ores containing zinc in much greater percentage than the ores of Joplin were heavily penalized and zinc was discarded wherever possible. Where zinc concentrates contained other valuable metals, such as lead, silver, or gold, these, if paid for at all, brought but a small return.

The electrolytic process, in conjunction with selective flotation, has in large measure solved the treatment of this class of ores. At the Salt Lake department of the Anaconda company, its subsidiary, the International Smelting Company, now pays for zinc instead of penalizing it, and by so doing has tremendously benefited and prolonged the life of the surrounding mining districts. Ores are purchased on a schedule which pays for zinc as well as lead. They are treated in the complex ore mill at Tooele, and lead, zinc, and iron-silver concentrates are made. The lead concentrate is treated in the local furnaces, the zinc concentrate, which contains considerable lead and silver, goes to the electrolytic zinc plants in Montana, where all three metals are recovered, and the iron-silver concentrate is used as flux. Thus a much greater variety of ores than formerly can be bought and better prices, of course, can be paid for them.

Another interesting problem which arose out of the zinc development was the recovery of that metal from lead blast-furnace slags, as is described elsewhere in this issue. Following many experiments in working out an idea proposed by U. A. Garred, a plant was built at East Helena which has been in successful operation over a year and is recovering about 80 per cent of the zinc contained in the formerly wasted slag. Not only is Anaconda benefiting by this development, but the smelter is receiving a substantial revenue from a material disposition of which formerly cost money. The process is a simple one and had no doubt occurred to many, but it remained for Anaconda to place it on the map. Coal dust and air in proper proportion are blown through molten slag. The coal reduces the zinc in the slag to metal, which volatilizes and at once reoxidizes in the atmosphere of the furnace and in a combustion chamber which follows it. The resultant leady zinc oxide is collected in a baghouse and sent to Great Falls for further treatment.

In Poland the company has large reserves of oxidized zinc ores. For their beneficiation a plant has been installed which employs a process developed by a German concern. This process also produces a zinc oxide from which metal is made by electrolysis. The zinc is volatilized by heating the ore together with about 30 per cent of its weight of coke breeze in a furnace similar to a cement kiln.

Another example of volatilization is the method developed at the Tooele plant many years ago for treatment of leady matte in the copper converters. This process has added greatly to the flexibility of the smelter, inasmuch as it makes possible the recovery of lead from blast-furnace matte, thus enabling the lead and copper departments of the plant to work together to better advantage.

**The city of Anaconda, with the smelter in the distance**

In the Andes mountains, on the edge of the Atacama desert, at a point called Potrerillos, about one hundred miles east of the little seaport town of Chanaral, in Chile, Anaconda owned a huge deposit of low-grade copper ore. The deposit contained both sulphides and oxides. Treatment of the former was a simple matter, as the sulphide minerals were readily recoverable by flotation in the form of concentrate containing 28 per cent copper. A mill for the concentrating of 15,000 tons of ore per day, together with roasting, smelting, and converting plants for the treatment of the concentrate resulting, was designed at Anaconda and erected at the property.

This part of the ore treatment required no departure from standard practice, and was therefore relatively simple and free from uncertainties. The case was different with respect to the oxide ore. A preliminary examination disclosed that this ore contained much iron and an unusually large percentage of arsenic, which dissolved in acid with the copper. To apply the electrolytic process to the foul solutions coming off the ore was out of the question. Not only would the current efficiency have been very low, but an unsatisfactory quality of copper would have resulted. After considerable experimentation with the old Van Arsdale process for precipitating metallic copper from hot solutions under pressure by means of sulphur dioxide (a method, by the way, which yielded good results and seemed quite practicable), it was finally decided to take a leaf out of the book of zinc and apply purification to the copper solutions before electrolyzing. A series of continuous experiments lasting a year was, therefore, carried through at Anaconda to develop a suitable process and collect sufficient data for the design of a purifying plant.

The process adopted consists of adding powdered limestone in carefully controlled amount to the strong copper solution as it comes from the leaching plant. If this be accurately done, iron, alumina, and arsenic, as well as a less common but decidedly harmful impurity, molybdenum, which occurs in minute quantity, are quite completely precipitated, with the simultaneous loss of only very little copper. If the addition of limestone be accomplished properly, and if the solutions have the right temperature, the slimy precipitate may be filtered fairly easily. The process requires careful control, but is commercially feasible, as has now been demonstrated in the plant at Potrerillos. Solutions which, without purification, would have yielded less than 16 lb. of copper per kilowatt-day, yield 28 lb., and the quality is of the highest. Apart from this step, the design of the oxide ore treatment plant follows more or less standard lines, but the inclusion of the purification step in the plant's flow sheet has made possible the treatment of excess wash waters in a novel manner, and has made it possible to turn out all copper as electrolytic.

It would be tedious and quite beyond the scope of the present paper to describe or even enumerate the many investigations that have been and are constantly being made, some bringing results, others only disappointment, or perhaps, by pointing the way to the solution of some other wholly disconnected problem, serve their purpose in a different manner. Some one has said that "Thoughts Are Things," and nowhere is that better illustrated than in the metallurgical business. Never have men of vision had a better opportunity to put into concrete form their dreams than by making themselves part of a modern business enterprise.

Life offers few greater thrills than to see one's thought take form in brick and steel. From vague and indefinable beginnings, these gradually, by dint of mental concentration and of well-directed research, take definite shape, and, in the form of processes, machines, and plants, take their place among the things that lighten the burdens of humanity and make life broader and more abundant.

## Anaconda's Subsidiaries

Anaconda Copper Mining Company was incorporated in Montana, June 18, 1895. The following are among the important subsidiaries that are owned outright or that are controlled through ownership of a majority of the stock:

International Smelting Company, with plants at Tooele, Utah, and Miami, Ariz.; International Lead Refining Company, East Chicago, Ind.; Raritan Copper Works, Perth Amboy, N. J.; United Metals Selling Company; American Brass Company; Anaconda Lead Products Company; Anaconda Wire & Cable Company; Andes Copper Mining Company; Chile Copper Company; North Lily Mining Company; Silesian-American Corporation (Giesche Spolka Alcyjna); and Butte Copper & Zinc Company (leased).

Anaconda also holds an important stock interest in Inspiration Consolidated Copper Company; Greene Cananea Copper Company; and Walker Mining Company.

**An aerial view of Butte, where mine and city meet**

# Selective Flotation of Lead-Zinc Ores at Tooele, Utah

By A. B. YOUNG and W. J. MCKENNA

*Assistant Manager and Concentrator Superintendent, respectively, International Smelting Company*

AMONG THE FIRST, if not the very first, to apply the principles of lead-zinc selective flotation to the treatment of ores bought on a purely custom basis was the Tooele, Utah, plant of the International Smelting Company. The early part of the present decade saw lead-silver mining in the Utah district seriously hampered by the ever-increasing quantities of zinc in the sulphide ores on the one hand, and silica in the oxidized ores on the other, while metal prices, particularly after the expiration of the Pittman Act, were at a low ebb. At the same time the iron fluxing situation was difficult. Iron is required for the elimination of both silica and zinc in lead smelting, and most of the iron flux available itself carried considerable zinc.

A. B. Young

Such in a broad way was the nature of the problem confronting the local staff of the company after the war. The first step was the scrutinizing of the local ore supply, the employment of a geological department, and the establishment of an ore-testing laboratory. Painstaking study showed that by selective flotation a large tonnage of byproduct zinc in ore already mined could be made available for the parent company's electrolytic plant at Great Falls, and that the bringing into production of quantities of lower-grade ores, hitherto not commercial, was highly probable. The second step was the remodeling of a copper concentrator, which had come into the possession of the company, for use in lead-zinc work. Before entering into the description of the concentrator and its operation, a word will be said concerning the economic benefits to the mining industry brought about through this operation and other similar enterprises.

First: Zinc, the bugbear of the lead smelter man (the increased cost incurred in lead smelting is fairly accurately represented in the customary penalties in lead ore schedules) is removed and put into a concentrate from which it is commercially recovered.

Second: Silica is removed. In many of the ores treated this consideration outranks the first, economically.

Third: Iron sulphides, with precious-metal content, are removed, these forming a cheap source of iron for the fluxing of other siliceous ores which cannot be so handled.

The combination of these factors has made it possible for ores to yield many more dollars per ton to their producers than would otherwise have been possible.

Whether it was better to try to induce the mines to install separate concentrators at their own properties or to establish a custom milling plant was a question. The following factors entered into the decision: First: Laboratory work had proved that ores from various mines could be milled in combination with as great ease as separately, within limits that will be discussed later. The apparent disadvantage in trying to mill two or more ores together was more imaginary than real, and in some important instances actual improvement has resulted from a combination of several ores. Second: Amortization would be less for a custom mill. A centralized mill would have a whole district to consider in amortization allowance, whereas a mill built for a single property having a limited ore reserve would have to be amortized quickly. Third: Securing of reasonable railroad freight rates on ores. The railroads early indicated their willingness to give rates on crude ore which would return to them no more money than the freight received on the concentrates from such ores. Of course, there are limits to this principle, but it was soon found that these limits are almost as extensive as the boundaries which control the freight situation on ores to be smelted. Fourth: A centralized mill operating in conjunction with a custom smelter could arrange its practice and classify its products so as to give the smelter the most advantageous fluxing charge at all times. Fifth: Conditions would be more favorable in a custom mill for low operating costs and improved metallurgical practice. A large centralized mill obviously can make much lower operating costs than several mills of smaller size, and likewise a central mill connected with a large smelter is undoubtedly in position to furnish better laboratory facilities and a more elaborate technical staff. It is in a much better position to assume the risks, or what we might call hazards of process, than it is possible for the individual mine operator, with his limited resources, to be.

W. J. McKenna

Hence, the time was quite ripe for the installation of a central custom mill when, in April, 1924, the International Smelting Company came into the possession of a concentrator of 1,000 tons' daily capacity, designed for treating copper ores. This mill consisted of a coarse-crushing plant, grinding machinery, and Callow cells, which will be described later, together with the necessary blowing equipment, two Oliver filters, and thickeners for concentrate and tailing. Investigation by the Geological Department of the company, and research on a multitude of samples of lead-zinc ore from Utah, southern Idaho, eastern Colorado, eastern Nevada, and to a lesser extent northern Arizona, showed that sufficient tonnage was available to warrant a 500-ton lead-zinc unit. A survey of the copper concentrator, which could not be operated economically at the low prices of copper prevailing, showed that by spending a few hundred thousand dollars it could be revamped for the treatment of lead-

zinc ores. The crushing and grinding equipment was already in; though, as a matter of fact, the custom ore would have to be put through the company's sampling mill, which of course took care of the coarse crushing. Remodeling included a rearrangement of the Callow cells to make room for more cells for the iron circuit. Sufficient flotation equipment was installed to handle a thousand tons of copper ore with one pass, but in treating lead-zinc ores three circuits of flotation were necessary—lead, zinc, and iron—the last being necessary because of the association of precious metals with the pyrite. The principal sources of ore supply were the Park City and Bingham districts, Park City being ready to ship at once.

The necessary appropriations were secured in the summer of 1924, and on Nov. 1, the first unit, capable of handling 500 tons a day, was placed in operation. Test work had shown that Bingham ores required slightly different treatment from those of Park City, so that separate runs were made on these two ores in this first unit. Shortly, however, the tonnage, particularly from Bingham, was increased so much that another 500-ton unit was necessary, which was placed in operation May 1, 1925. Since then the concentrator has operated on a two-section basis, each having a daily capacity of 500 to 600 tons.

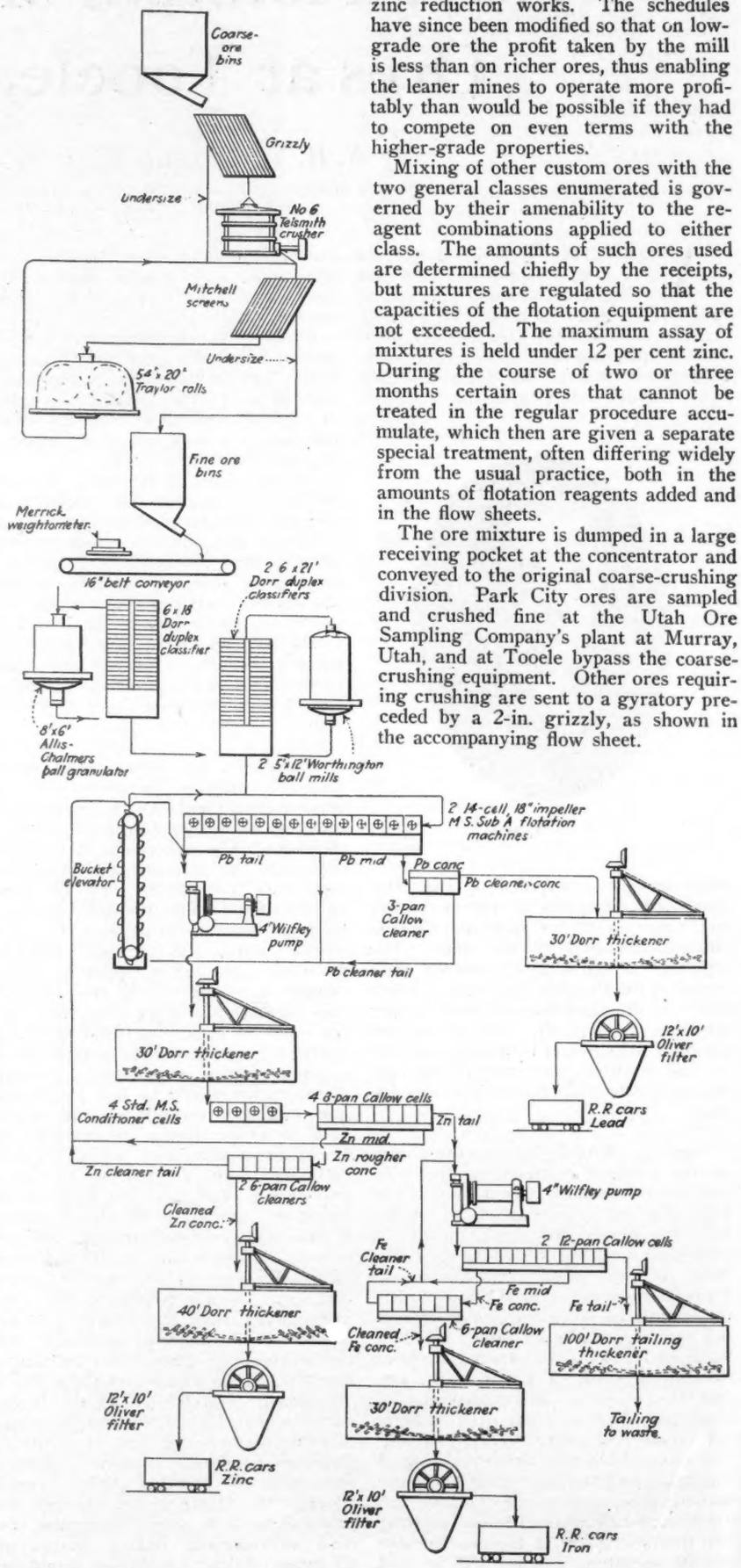
THE lead-zinc ores treated here fall into two general classes, the Park City type and the Bingham type, though a few ores from Park City respond to the Bingham type of treatment better than to their own. Park City ores generally are coarsely crystalline, contain a lower proportional amount of pyrite, and purer sphalerite; Bingham ores are more finely disseminated and contain more iron and are less amenable to selective flotation.

Prior to the introduction of selective flotation on a custom basis on a large scale in the Utah district, the miner usually shipped his ore to a mill owned by another, where it was treated by itself. The miner paid a toll charge to the owner of the mill and the concentrates were returned to the original owner for sale to the smelter or other reduction plant. Such a procedure was altogether too cumbersome from a business standpoint; and the International staff having a decade or so of ore purchasing experience behind it, realized that the preferable way was to conform to customary smelter practice in ore purchasing and to evolve a system of schedules for the actual purchase of the ore from the miner, the mill products to be the property of the smelting company. With the background of experience it was relatively easy to develop a purchasing schedule that would yield the Smelting company a fair profit and also give to the ore producer a quick and accurate determination of the value of his product. The schedules were framed so that the mill profited on the lead and silver, was protected against loss in gold, but made no profit on zinc, all margin on that metal being left to the zinc reduction works. The schedules have since been modified so that on low-grade ore the profit taken by the mill is less than on richer ores, thus enabling the leaner mines to operate more profitably than would be possible if they had to compete on even terms with the higher-grade properties.

Mixing of other custom ores with the two general classes enumerated is governed by their amenability to the reagent combinations applied to either class. The amounts of such ores used are determined chiefly by the receipts, but mixtures are regulated so that the capacities of the flotation equipment are not exceeded. The maximum assay of mixtures is held under 12 per cent zinc. During the course of two or three months certain ores that cannot be treated in the regular procedure accumulate, which then are given a separate special treatment, often differing widely from the usual practice, both in the amounts of flotation reagents added and in the flow sheets.

The ore mixture is dumped in a large receiving pocket at the concentrator and conveyed to the original coarse-crushing division. Park City ores are sampled and crushed fine at the Utah Ore Sampling Company's plant at Murray, Utah, and at Tooele bypass the coarse-crushing equipment. Other ores requiring crushing are sent to a gyratory preceded by a 2-in. grizzly, as shown in the accompanying flow sheet.

## Flow Sheet of the Concentrator



The fines and the crusher product are screened by two 4x6-ft. Mitchell vibrating screens equipped with 1¼-in. mesh screen. Oversize from the screens goes to a set of 54x20-in. heavy-duty Traylor rolls in closed circuit with the screens. Undersize is conveyed to one of the two fine-ore bins, each ahead of a 500-ton mill section. Each bin has a capacity of 650 tons.

The mixed ores for each section, crushed to —1¼ in., are fed from separate bins to the grinding equipment, which for both sections is identical, a ball mill in closed circuit with a classifier. The classifier overflow is equally distributed to two secondary classifiers, each in closed circuit with a ball mill. The secondary classifier overflow forms the flotation feed. Manganese steel liners are used exclusively in the primary ball mills and hard white iron liners in the secondary ball mills. Regular additions of 4-in. forged steel balls are made to the primary mills, and 2-in. cast-iron balls to the secondary mills. Pulp densities in the primary mill are held as closely as possible to 72 per cent solids; in the secondary at 68 per cent. The density of the secondary classifier overflow, or flotation feed, ranges from 22 to 24 per cent solids. A typical screen analysis follows:

| Screen Size | Per Cent Weight of Total |
|---|---|
| Plus 65 | 1.6 |
| Plus 100 | 5.6 |
| Plus 150 | 7.6 |
| Plus 200 | 14.8 |
| Minus 200 | 70.4 |
| | 100.0 |

Two types of flotation equipment are in use at this plant: Minerals Separation Sub-Aëration machines, and Callow pneumatic cells. The first named are used as roughers in the lead circuits of both sections and as roughers in the zinc circuit of one section, whereas the pneumatic cells are used as roughers in the zinc circuit on one section, as roughers in the iron circuits of both sections, and as cleaners in the lead, zinc, and iron circuits of both sections. The M. S. machines are of the standard 18-in. full double disk agitator type, each having 14 cells. All of the Callows are of the old 3x3-ft. pan type. The flow sheet on p. 292 shows flotation Section 1. Section 2 is similar except that the pulp from the 30-ft. thickener is not conditioned but goes direct to two 14-cell M.S. Sub-A. machines. Zinc rougher concentrate is cleaned in two 3-pan Callow cleaners and zinc tailing is floated in two 8-pan Callow cells instead of the 12-pan equipment in Section 1.

The general practice followed in each circuit is to take a concentrate and a middling from each rougher. Rougher concentrate is cleaned, and cleaner tailing, together with rougher middling, pumped or elevated back to the head of the circuit. In no case is cleaner tailing or middling reground in a ball mill. It will be noted by the accompanying flow sheet that surge tanks are installed between the lead and zinc circuits of both sections. These tanks, which are 30x10-ft. Dorr thickeners, serve to increase the time of contact for flotation reagents and afford better density control in the zinc circuits. The flow sheet of Section 1 includes a conditioner between the surge tank and Callow zinc rougher cells. This four-cell conditioner is of the standard Minerals Separation type, equipped with 24-in. cruciform agitators. Air for flotation is furnished by two belt-driven No. 6½ Roots blowers. The air is generated at a pressure of 4½ lb. per square inch, this being the necessary pressure for the Callow equipment; the pressure is reduced to 1 lb. per square inch for the Sub-A. machines.

STANDARD reagent combinations are used such that a fairly clean separation is made between the three major minerals, galena, sphalerite, and pyrite. The reagents include soda ash, thio-carbanilid, sodium cyanide, zinc sulphate, potassium ethyl xanthate, copper sulphate, Aërofloat, and steam.

*Soda ash* is used as an alkaline conditioning reagent in the lead and zinc circuits of both sections. Its addition to the ores of the Bingham type is mostly with the feed to the ball mill in amounts ranging as high as 5 lb. per ton of ore. Addition of more soda ash to the zinc circuit is seldom required for this type of ore. The ores of the Park City type require little, if any, soda ash added to the lead circuit, but do require from two to three and a half pounds per ton in the zinc circuit.

*Thio-carbanilid* is used as the basic collector in the lead circuits of both sections and is added directly to the primary ball mills at the rate of 0.10 lb. per ton. Its use, although not absolutely necessary, tends to stabilize collecting conditions and increase the recovery of lead without noticeably affecting selectivity in the zinc circuits.

*Sodium cyanide* is used as a lead accelerant and as a zinc-iron depressant. Its use is always in conjunction with zinc sulphate and it is added to the primary ball mill. The Park City ores generally require 0.3 to 0.4 lb. sodium cyanide per ton; the Bingham ores, although having a higher iron content, require less—0.20 to 0.30 lb. per ton.

*Zinc sulphate* is used as a depressant in conjunction with sodium cyanide and is added in solution to the primary mills of both sections. The consumption for the Bingham and Park City types of ore ranges from 0.6 to 1.2 lb. per ton of ore.

*Potassium ethyl xanthate* is used as an auxiliary collector in the lead and zinc circuits and is the only collector added to the iron circuits. Of the total amount of 0.2 to 0.3 lb. per ton added to the lead, zinc, and iron circuits, only 15 per cent is usually added to the lead and zinc circuits. This reagent has been found very beneficial, but must be held to a minimum to insure selectivity.

*Copper sulphate* is used as an accelerant in the zinc circuits, being added in crystalline form to the lead rougher tailing. The amounts added are dependent upon the character of the ores treated and the condition of the circuit as influenced by other reagents. The addition of copper sulphate ranges from 0.7 to 1.5 lb. per ton.

*Aërofloat or phospho-cresylic acid* is used as the basic collector and frother in the zinc circuits of both the Bingham and Park City ores, being added to the lead tailing. The addition of 0.15 lb. per ton is fairly constant for both types of ores.

*Steam* is added to the lead rougher tailing ahead of the zinc circuits of both sections, and the pulp heated to 30 deg., resulting in a marked improvement in the grade and recovery of zinc.

As yet, no successful chemical controls have been developed to govern the amounts of reagents added. Titrations for soda ash alkalinity and *pH* determinations have not shown anything tangible for actual operating. Control of the three circuits is therefore entirely visual, based upon: First, the character and appearance of froths; second, the use of the vanning plaque; and third, microscopic observation of products. The flotation operators have become very efficient in handling the three different circuits by the methods detailed in the foregoing, and the changing of reagents is left more or less to their judgment. It has, however, become necessary to establish limits on the amounts of certain reagents, as they interfere with selectivity if too concentrated in the return circuit water.

Assays of the flotation feed and products of each section, typical of recent concentrator operations, are shown in the table at the bottom of this page.

All three thickeners used for handling concentrates are of the regular Dorr type. The tanks are constructed of redwood and are 30 ft. in diameter by 10 ft. high. Zinc concentrates from both sections are combined after separate sampling and partly dewatered in one thickener prior to filtering. Lead and iron

**Assays of Feed and Products**

| | Section I | | | | | | Section II | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Per Cent Pb | Oz. Ag | Oz. Au | Per Cent "Insol." | Per Cent Fe | Per Cent Zn | Per Cent Pb | Oz. Ag | Oz. Au | Per Cent "Insol." | Per Cent Fe | Per Cent Zn |
| Feed | 2.9 | 8.33 | 0.10 | .. | 9.3 | 10.3 | 8.6 | 5.12 | 0.05 | ... | 10.9 | 7.1 |
| Lead concentrate | 65.3 | 140.61 | 0.787 | 2.0 | 3.3 | 7.1 | 66.4 | 35.57 | 0.10 | 2.4 | 2.4 | 5.6 |
| Zinc concentrate | 0.9 | 7.10 | 0.10 | 1.9 | 2.7 | 59.1 | 2.7 | 2.16 | 0.04 | 1.9 | 1.6 | 59.8 |
| Iron concentrate | 1.5 | 6.64 | 0.16 | 8.6 | 36.9 | 4.3 | 5.0 | 6.16 | 0.14 | 9.3 | 34.8 | 2.7 |
| Final tailing | 0.2 | 1.53 | 0.045 | ... | 8.3 | 0.4 | 0.4 | 0.58 | 0.025 | ... | 10.0 | 0.2 |

concentrates are handled similarly. However, provisions have been made to thicken any desired amount of iron concentrate with the lead concentrate, and the excess separately. This feature gives a flexible method for distributing iron concentrate to either the copper or lead plant, for fluxing purposes. For thickening final tailing two Dorr thickeners are provided—one 100-ft. and one 60-ft., but ordinarily only the larger is used for both 500-ton sections. The lead, zinc, and iron concentrates are thickened to 50 per cent solids before going to the filters. Vacuum for the filters is furnished by Ingersoll-Rand dry vacuum pumps.

Fresh water sources are several mountain springs about three miles away, from which the water is piped to a concrete storage reservoir of 412,000-gal. capacity. A partial analysis follows:

**Analysis of Fresh Water**

| | Grains per U. S. Gallon |
|---|---|
| Total solids | 32.67 |
| Organic matter | 7.47 |
| $SiO_2$ | 0.58 |
| $Fe_2O_3$ and $Al_2O_3$ | 0.35 |
| CaO | 6.88 |
| MgO | 3.73 |
| $SO_3$ | 5.13 |

A high content of calcium and magnesium salts makes the water very hard. The supply of fresh water is so limited that its addition to the three circuits is only in amounts sufficient to make up for losses in thickened tailing and the moisture contained in concentrates. Overflows from all surge tanks, and concentrate and tailing thickeners are combined and pumped to the return circuit equalizing tanks above the mill. This water is used in the lead, zinc, and iron circuits, and as floor wash water throughout the mill. The net new water used by the mill amounts to 17.5 g.p.m. per 100 tons of ore treated daily.

THE ore-testing laboratory has been an important factor in the development of processes and in actual operation. In a general way the work of the laboratory includes that done in connection with ore purchasing, service to outside operators, and plant and research work.

Samples sent to the Ore Purchasing Department from prospects and producing mines are tested in the laboratory for their amenability and metallurgical results. The batch testing follows standardized flotation practice in current use at the concentrator relative to reagent combination and flow sheets. Results of these tests, which are accurately correlated with concentrator results both as to grades of products and recoveries, form a basis for determining schedules in the purchase of the ore represented. For correlating laboratory and concentrator results, factors are determined monthly by testing samples of the various ores making up the concentrator feed.

A number of operators have received the benefit of the service work done on samples of their ores, not only lead-zinc but also copper and silver. Much assistance is given in improving the smaller operator's metallurgical results.

The plant work covers the constant testing of all ores received at the plant for concentrator treatment, so that mixtures are made according to the indicated amenability of the ores to the standard process. Also, considerable research work is done with new reagents and with processes for the improvement of present practices.

Briefly, standard batch flotation testing in the laboratory follows the grinding of 600-gram samples of ore in a small laboratory ball mill at a density of 1:1—together with any reagents such as soda ash, thio-carbanilid, or xanthate. The ground pulp is removed from the ball mill and transferred to a Fahrenwald laboratory flotation machine, where it is further diluted to 22 per cent solids. Necessary reagents are added, and the resulting lead, zinc, and iron concentrates obtained. These products and the tailings are filtered, dried, and weighed, and results calculated from their assays.

In conjunction with the ore-testing laboratory is a small continuous test mill operated by the laboratory staff. Its capacity is 100 lb. per hour, with flotation equipment of the same types as are in use at the concentrator. It was designed to duplicate regular concentrator practice in every important detail, including grinding, flotation, thickening, and the use of return circuit water. The test mill is seldom used to test the amenability of ores, but is of importance in the application of new processes and new flotation reagents. By its use, wider ranges of processes and reagent combinations can be tried without the losses that might be experienced if a section at the concentrator were used.

Great and many have been the improvements in mill practice at the Tooele lead-zinc concentrator since it began operation in the latter part of 1924. Recoveries have been increased, grades of products raised, and operating costs lowered, with the result that increased profits have been brought about. Some of these increased profits have been retained by the company, but a great portion has been passed back to the shippers in one way or other, thus giving the industry as a whole the benefits of such improvements. These increased profits have allowed shippers to mine lower grades of ore, the grades treated now being only half of what they were when operations began. This must always be the trend in mining: continued metallurgical improvements enabling the extraction of metals from ores to be done more cheaply all the time. Otherwise the world's metalliferous deposits would be depleted far more rapidly than now.

The results obtained in lead-zinc selective flotation here have not been achieved entirely by the metallurgists immediately on the job. The railroads, for instance, have contributed, with the establishment of freight rates permitting the ores and products to move. The chemical houses, with their research work, have been bringing out new reagents, each a little better than its predecessors. It is a far cry from the chemicals we had to work with in the period from 1920 to 1923, to the thio-carbanilid, xanthate, and Aërofloat of today. Furthermore, great help has been given by the U. S. Bureau of Mines and the University of Utah, both of which institutions have pioneered the true research work of this kind, pointing the paths that practical operators have followed. Nor should the junior staff of our company, who have not only had the brunt of the actual test work to bear but who have given innumerable new ideas to the process, go unmentioned. Above all, the custom lead-zinc selective flotation industry which has been built up in this district could not have been successful had we not had the backing and confidence of the higher officials of the company who have had the faith to go through with us in the pioneering of a rather unknown field, instead of allowing the whole program to lapse into a laboratory dream.

**Part of mill feed comes from Bingham Canyon by aërial tram**

# Both Copper and Zinc Ores Treated by Selective Flotation

## in Concentrators at Anaconda, Montana

By BAYARD S. MORROW
*Superintendent of Concentration*

BEFORE DESCRIBING the selective flotation practices now in use at Anaconda for the treatment of copper and zinc ores, a brief history of early concentrating practices at Anaconda will be in order, as a measure of the progress that has been made. The original concentrating plants were erected for the treatment of copper sulphide ores, with much iron, both as pyrite and in association with the copper sulphides, and also carrying silver and gold associated principally with the copper minerals. No zinc ores were concentrated at Anaconda prior to the year 1916.

The first concentrator was constructed in 1883 and 1884, going into operation in October, 1884. It was located at what was termed the Upper Works, one mile north of the town of Anaconda, and, as originally constructed, utilized Blake crushers as primary and rolls as secondary breakers; and trommels for sizing at 3, 5, 6, 8, and 10 mesh, the various sizes being jigged in Harz jigs. No provision was made at first for concentrating the fine material, this being added to the jig concentrate and smelted. Later, hydraulic classifiers and Frue, Embrey, and Triumph vanners were added for concentrating the slime. The original capacity was about 1,000 tons per day.

During 1886 steam stamps were added to the crushing equipment and Collom jigs and revolving round tables to the concentrating equipment, bringing the capacity of the plant to about 2,000 tons per day.

About 1887 the Lower Works concentrator was erected at the now extinct town of Carroll, two miles east of the Upper Works. The capacity was 1,750 tons per day. Steam stamps and Heberle grinders constituted the crushing equipment, and trommels and shallow pocket classifiers were used for sizing and classifying the feed to Collom and circular jigs. Double- and single-deck round tables constituted the slime-concentrating equipment. Later the circular jigs were replaced by Collom jigs; Harz jigs were introduced ahead of the stamps in a portion of the mill for the treatment of original fines from the so-called "soft ores"; and the Heberle grinders were replaced by Cornish rolls.

The appearance of the Wilfley table in 1896 resulted in its installation in the

Lower Works concentrator and in the erection of the Wilfley table plant at the Upper Works. Chilean mills were also added at this time, 1896 to 1900. I am indebted to an article by the late C. W. Goodale (*Trans.* A.I.M.E., Vol. 26, p. 559) for the greater part of this history.

In 1901, the present concentrator buildings were erected on the site of the New Works—i.e., the present Anaconda Reduction Works. The original concentrator went into operation during February, 1902, and consisted of two mills with a steam power plant between them. There were eight duplicate units, four in each mill, each having a capacity of 1,000 tons per day, or a total daily capacity of 8,000 tons.

An installation of Blake crushers, in two stages, constituted the coarse-crushing, and rolls, in three stages, the fine-crushing equipment. Sizing was done with trommels at 1.25 and $\frac{7}{8}$ in., and 7, 5, 2.5 and 1.5 mm. Sizes between 1.25 in. and 7 mm. were concentrated on Harz jigs and between 7 mm. and 1.5 mm. on Evans jigs, and all jig tailings coarser than 1.5 mm. were returned to the various roll circuits for further crushing, sizing, and jigging. Minus-1.5-mm. material was hydraulically classified, the spigot products concentrated on Evans jigs, the overflows deslimed in V-shaped deslimers, and the deslimed portion concentrated on Wilfley tables. The middling from the jigs treating minus-1.5-mm. material was ground in Huntington mills to minus-1.0 mm. and again concentrated on Evans jigs. Tailings were rejected from the 1.5-mm. jigs, Huntington mills jigs, and the Wilfley tables. The slime from the V-shaped deslimers was impounded in ponds and settled, the settlings then being excavated and briquetted for subsequent smelting in the blast furnaces which were then used instead of the reverberatories of today.

Between 1902 and 1915 the concentrator capacity was increased to 1,500 or 1,600 tons per unit per day by the substitution of larger crushers, rolls, and elevators, and by the addition of trommels, jigs, and tables. During 1912 one unit (No. 1) was remodeled for the use of Hancock jigs for concentrating plus-$\frac{3}{8}$-in. material and Wilfley tables for the finer sizes. The jig tailings were crushed to table size and there concentrated. During 1913 a slime plant, consisting of shallow Dorr thickeners and 20-deck round tables, was erected. This plant included Dorr concentrate thickeners and Oliver filters for dewatering the slime concentrate, and operated as a round table plant from April, 1914, for about four years, or until about February, 1916, when a far superior method was discovered.

During 1915 the flotation process was adopted as an adjunct to gravity methods. The crushing, sizing, and jigging units down to 1.5 mm. were used with little or no change in arrangement. The minus-1.5-mm. material was deslimed in hydraulic deslimers, the spigot product concentrated on Wilfley tables, and the slime thickened in the existing slime thickeners. Table tailing was ground in ball mills operating in closed circuit with Dorr classifiers and floated in standard Minerals Separation flotation machines. The thickened slime was floated in a separate plant, also using standard M.S. machines for the purpose. The original flotation circuits were arranged for the collective flotation of the sulphides, and between 1915 and 1927 progressed through the following cycles.

1915 into 1923: Collective float, acid circuits with collecting and frothing oils.

1923 into 1924: Collective float, acid circuits with xanthate as the collector and frothing oils.

1924 into 1927: Collective float, alkaline circuits with xanthate as the collector and pine oils as the frothers.

Hydrated lime in the form of milk of lime was used as the alkaline agent. No changes in equipment were found necessary at the mill for the alkaline circuit. At the slime plant a few of the M.S.

machines were converted into agitating conditioners for the purpose of preagitating the slime pulp with lime, and a Callow flotation plant was added. Flotation concentrates were dewatered in Dorr thickeners and Oliver filters.

During the first half of 1926, exhaustive tests were undertaken to develop the possibilities of applying selective flotation to the separation of the various copper sulphides from the pyrite and gangue. The copper-bearing ores are sulphide ores from the Butte mines and contain much iron as pyrite, as well as iron combined with copper as bornite and chalcopyrite. The principal copper minerals are: Chalcocite, $Cu_2S$; enargite, $Cu_3AsS_4$; bornite, $Cu_5FeS_4$; chalcopyrite, $CuFeS_2$ or $Cu_2S \cdot FeS_2$; covellite, CuS; tetrahedrite, $4Cu_2S_3 \cdot Sb_2S_3$; and tennantite, $Cu_3As_2S_7$. Silver is associated principally with the copper minerals, but a small amount of gold is associated with both the copper sulphides and with the pyrite. The gangue is quartz, highly altered granite, and feldspar, with sphalerite and galena and more rarely barite and hübnerite as accessory minerals. A typical analysis of the present mill heads is:

| | Per Cent | Oz. per Ton |
|---|---|---|
| Copper | 4.20 | .... |
| Silver | .... | 2.00 |
| Gold | .... | 0.010 |
| Iron | 11.0 | ..... |
| Sulphur | 13.3 | ..... |
| $SiO_2$ | 57.7 | ..... |
| $Al_2O_3$ | 7.7 | ..... |
| CaO | 0.6 | ..... |

Zinc, lead, arsenic, and antimony are also present in small amounts.

A cursory examination of the minerals present and of the ore analysis indicated that the application of selective flotation would entail not only an exhaustive investigation, but would in all probability open up a grinding problem that might require marked changes in equipment. A series of laboratory tests indicated the possibility of producing a concentrate assaying 20 per cent copper, compared to 11 per cent for the combined gravity and collective flotation concentrate, without an abnormal increase in the copper content of the tailing. The tests also indicated that the standard M.S. machines of Anaconda design were not well adapted to selective practice and that either a subaërated or pneumatic type machine would be more suitable.

Early in 1926 the investigation was transferred from the laboratory to the mill proper. The slime flotation plant was equipped with Callow flotation machines (pneumatic type), so the entire slime plant operations were changed from collective to selective flotation with no change in equipment. This was accomplished by increasing the alkalinity of the pulp from barely alkaline to an alkalinity of 0.3 lb. of free calcium oxide per ton of tailing water, and by reduction of xanthate from 0.75 to 0.20 lb. per ton of slime treated. The frothing oils remained practically unchanged in quantity and in kind.

Anaconda slime, although containing practically the same amount of copper as the original ore, has but about half the iron; therefore, the production of a concentrate assaying 20 per cent copper did not call for the rejection of any excessive amount of pyrite. Owing to the colloidal alumina and silica present, the production of a concentrate assaying much in excess of 20 to 22 per cent copper results in a sharp increase in tailing losses and so has not been attempted. The transfer from collective to selective flotation of Anaconda slime was accomplished in less than thirty days. The metallurgical results obtained at this time are typified in Table I.

The deslimed ore had approximately the same copper content as the slime, but contained about twice as much iron, owing to the presence of the bulk of the pyrite from the original ore.

Investigation at this point was carried out on a scale of from 200 to 300 tons per day, using a portion of one of the idle mill units for the purpose. One standard 24-in. M. S. machine was remodeled into a 21-in. Sub-A. type, and an experimental Callow unit consisting of a rougher and cleaner installed. Metallurgical results were about equal between the two types of machines when the Callow blankets were kept in perfect condition, but owing to the relatively large amounts of burned lime required for proper pyrite rejection, the Callow blankets blinded rapidly and 4½-to-5-lb. air pressure was required. Blanket upkeep was high, and blanket conditions caused erratic metallurgical results. The M. S. Sub-A. machine proved well suited to the work, producing a finished concentrate and a final tailing from fifteen cells at a rate of 220 to 250 tons per day without the use of a separate cleaner. Relatively fine grinding 8 per cent plus 65 mesh and 25 plus 100 mesh, was necessary to maintain the grade of concentrate and to prevent undue tailing losses.

During the test period a small amount of gravity concentrate, high in iron and low in "insoluble," was produced on Wilfley tables prior to flotation, in anticipation of requirements for a concentrate of this character for acid manufacture and for fluxing converter slags. Typical metallurgical results after the details of manipulation and practice had been solved are exemplified by the tabulation in Table II.

The investigation, as carried out during the first half of 1926, indicated that the selective flotation of Anaconda slime and virtually self-supporting—that is, that the reduction in reagent costs, and the reduced amount of concentrate to handle, practically balanced the increased losses of copper, gold, and silver in the slime tailing. It also indicated that selective flotation of deslimed ore warranted the remodeling of one concentrator unit for selective practice in order to secure sufficient concentrate to conduct roasting and smelting tests on a commercial scale. Accurate data were desired as to the smelter's ability to overcome the additional tailing losses at the concentrator.

One unit was, therefore, remodeled, utilizing the existing crushing and grinding equipment, but bypassing the jigs and tables to five Sub-A. machines, which occupied the space formerly taken by four standard spitz type machines.

**TABLE I—Slime Assays**

*Base metals in per cent; gold and silver in ounces per ton*

| | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
|---|---|---|---|---|---|---|---|---|
| Feed | 4.38 | 2.8 | 0.008 | 6.3 | 8.0 | 55.7 | 16.5 | 0.8 |
| Concentrate | 19.5 | 12.5 | 0.030 | 20.3 | 30.4 | 14.1 | 4.6 | 0.8 |
| Tailing | 0.36 | 0.33 | 0.001 | 2.5 | 2.0 | 67.2 | 17.8 | 0.7 |

**TABLE II—Assays of Feed and Products**

*Base metals in per cent; gold and silver in ounces per ton*

| | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
|---|---|---|---|---|---|---|---|---|
| Feed | 4.5 | 2.8 | 0.012 | 12.0 | 15.4 | 53.2 | 7.6 | 0.7 |
| Gravity concentrate | 10.0 | 5.1 | 0.017 | 32.0 | 38.9 | 11.8 | 0.7 | 0.2 |
| Flotation concentrate | 22.0 | 10.5 | 0.038 | 25.0 | 36.0 | 6.6 | 1.5 | 0.7 |
| Tailing | 0.23 | 0.16 | 0.001 | 100.0 | ..... | 66.2 | 7.0 | 0.5 |

**TABLE III—Assays With Revised Flow Sheet**

*Base metals in per cent; precious metals in ounces per ton*

| | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
|---|---|---|---|---|---|---|---|---|
| Ore (mill feed) | 4.1 | 2.2 | 0.011 | 11.0 | 12.8 | 54.6 | 8.8 | 0.5 |
| Table concentrate | 11.7 | 5.9 | 0.023 | 30.0 | 36.9 | 13.9 | 2.7 | 0.4 |
| Mill flotation concentrate | 23.6 | 12.5 | 0.050 | 25.2 | 36.9 | 5.1 | 2.7 | 0.5 |
| Slime flotation concentrate | 21.0 | 11.5 | 0.030 | 20.9 | 31.1 | 12.4 | 4.5 | 0.5 |
| Total flotation concentrate | 23.0 | 12.3 | 0.045 | 24.2 | 35.6 | 6.8 | 3.1 | 0.5 |
| Total concentrate | 21.0 | 11.1 | 0.041 | 25.2 | 35.8 | 8.1 | 3.0 | 0.5 |
| Mill tailing | 0.24 | 0.18 | 0.002 | 8.3 | 8.7 | 68.1 | 8.6 | 0.8 |
| Slime tailing | 0.40 | 0.25 | 0.002 | 2.3 | 2.1 | 65.1 | 18.3 | 0.8 |
| Total tailing | 0.27 | 0.20 | 0.002 | 7.0 | 7.5 | 67.5 | 10.4 | 0.8 |

**TABLE IV—Typical Assays**

*Base metals in per cent; precious metals in ounces per ton*

| | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
|---|---|---|---|---|---|---|---|---|
| Mill feed | 4.3 | 1.9 | 0.011 | 11.0 | 12.7 | 56.5 | 7.7 | 0.5 |
| Table concentrate | 11.2 | 5.4 | 0.025 | 31.0 | 38.9 | 11.4 | 1.6 | 0.4 |
| Mill flotation concentrate | 26.8 | 11.3 | 0.061 | 23.4 | 36.5 | 4.6 | 0.8 | 0.5 |
| Slime flotation concentrate | 19.9 | 9.3 | 0.032 | 20.2 | 30.2 | 14.3 | 4.9 | 0.5 |
| Total flotation concentrate | 25.1 | 10.8 | 0.054 | 22.6 | 35.0 | 7.1 | 1.8 | 0.5 |
| Total concentrate | 22.5 | 9.8 | 0.048 | 24.2 | 35.8 | 7.9 | 1.7 | 0.5 |
| Mill tailing | 0.22 | 0.14 | 0.002 | 8.5 | 8.9 | 67.3 | 7.9 | 0.7 |
| Slime tailing | 0.39 | 0.22 | 0.002 | 2.4 | 2.0 | 66.2 | 15.8 | 0.8 |
| Total tailing | 0.25 | 0.16 | 0.002 | 7.4 | 7.7 | 67.1 | 9.4 | 0.7 |

Tonnage was reduced from 2,000 to 1,500 tons per day in order to grind finer without any changes in the grinding circuit. This unit (No. 1) went into operation Dec. 1, 1926. The results obtained were even more satisfactory than the 200- or 300-ton investigation indicated. A full-scale roasting test in one furnace fed with a proper mixture of both flotation concentrates indicated that dust losses would not be prohibitive and that the calcine produced would be readily smeltable. By July 1, 1927, after further remodeling, facilities were provided to handle all of the ore being concentrated.

As remodeled, the former crushers, rolls, trommels, and their accompanying machinery were retained in their original locations, and the ball mills and classifiers were without change. Five M. S. Sub-A. machines replaced four standard machines, and the unit tonnage was reduced from 2,000 to 1,500 per day.

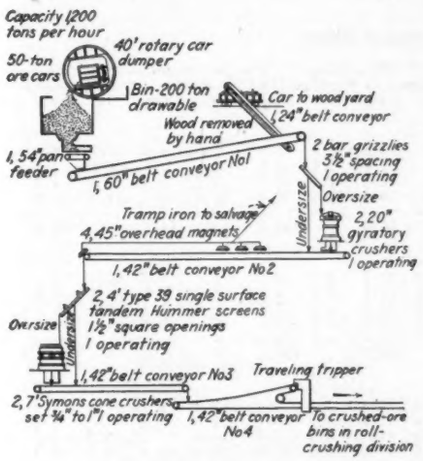

Capacity 1,200 tons per hour
40' rotary car dumper
50-ton ore cars
Bin-200 ton drawable
Car to wood yard
Wood removed by hand
24" belt conveyor
2 bar grizzlies 3½" spacing 1 operating
1,54" pan feeder
1,60" belt conveyor No1
Oversize
Undersize
Tramp iron to salvage
4,45" overhead magnets
2,20" gyratory crushers 1 operating
1,42" belt conveyor No2
2,4' type 39 single surface tandem Hummer screens 1½" square openings 1 operating
Oversize
Undersize
1,42" belt conveyor No3
Traveling tripper
2,7' Symons cone crushers set ¾" to 1" 1 operating
1,42" belt conveyor No4
To crushed-ore bins in roll-crushing division

**Fig. 1—Flow sheet of coarse-crushing division of copper concentrator, sections 2 to 8 inclusive**

Typical metallurgical results are contained in Table III.

During the summer of 1927 investigation disclosed that grinding to 5 per cent plus 65 mesh and 13 to 15 per cent plus 100 mesh would benefit both concentrate and tailing. Also the existing feed control by hand-operated bin gates was not sufficiently accurate for sound and uniform operating conditions in the ball mill and flotation circuits; and, because of the scattered arrangement of crushers, rolls, trommels, and other paraphernalia, it was not possible to reduce labor and other costs in proportion to the reduction in unit tonnage. Consequently, a high milling cost prevailed.

The abandonment of the jigs and tables made possible the realization of a long-cherished dream of a central crushing plant such as would permit further reduction by rolls or by ball mills if desired. So, during the latter part of 1927, such a plant, capable of crushing 10,000 to 12,000 tons of ore in two eight-hour shifts, delays included, was erected. This plant includes the ore-dumping equipment and its layout is made clear in Fig. 1. Ore-dumping equipment consists of one 25-ton Baldwin-Westinghouse push-pole locomotive for spotting railroad cars on one 40-ft. Wellman-Seaver-Morgan revolving car-dumper. The ore is first crushed in a gyratory set at 3½ in., and afterward in a cone crusher, with intermediate screening. The crushed-ore bins are in the concentrator building and serve seven of the eight units. One unit (No. 1) retains its former ore bin and flow sheet and is used for the treatment of rejects from the sample mill and for the production of a table concentrate suitable for the manufacture of sulphuric acid and for fluxing converter slag.

The fine-ore bin previously referred to is located in the area formerly occupied by the jaw crushers and coarse and intermediate rolls, together with their attending trommels, elevators, and launders. Jaw crushers for seven concentrator units have been eliminated, and the first two sets of rolls, with their trommels and elevators, have been moved into the area formerly occupied by the Evans jigs. The coarse and intermediate rolls are immediately back of and on the same level as the fine rolls and are close-circuited with ⅜-in. round hole trommels. The fine rolls are closed-circuited with 1½x12-mm. slotted trommels. All of the Anaconda deslimers have been moved from the rear to the front of the table floor and immediately back of the ball mills. Reference to Fig. 2 makes clear the flow of ore. The Hardinge mills (see Fig. 3)

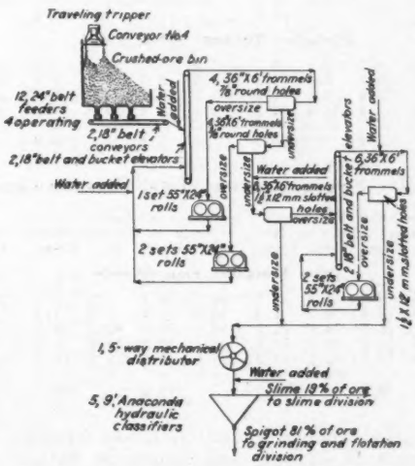

Traveling tripper
Conveyor No4
Crushed-ore bin
12, 24" belt feeders 4 operating
Water added
4, 36"x6' trommels 7/8" round holes
oversize
undersize
4, 36"x6' trommels 7/8" round holes
2, 18" belt conveyors
2, 18" belt and bucket elevators
Water added
1 set 55"x24" rolls
2 sets 55"x24" rolls
2 sets 55"x24" rolls
6, 36"x6' trommels
1½ x 12 m.m. slotted holes
1, 5-way mechanical distributor
Water added
Slime 13% of ore to slime division
5, 9' Anaconda hydraulic classifiers
Spigot 81% of ore to grinding and flotation division

**Fig. 2—Roll crushing and desliming division, sections 2 to 7 inclusive, one section shown. Capacity each section, 1,500 tons**

have been enlarged from 7 ft. 4 in. to 8 ft. 4 in. inside diameter and the 2-ft. Dorr classifiers replaced by 4-ft. Dorrs with a 48-in. depth of pool. The combined action of uniform feed rate, enlarged ball mills, and increased classifier capacity materially increased the fineness of grinding and has improved flotation metallurgy beyond expectations. Uniform conditions brought about by mechanical instead of hand-feed control have aided greatly.

Typical metallurgical results prevailing under the present flow sheet, Figs. 1 to 5, are exemplified in Table IV.

The principal factors governing the successful application of selective flotation to the copper ores treated at Anaconda are:

1. *Fineness of Grinding*—Grinding must be carried to the point where the major portion of the pyrite is freed from the copper sulphide so that pyrite rejection may not be accompanied by undue loss of copper.

2. *Proper Alkalinity Control*—Pulp alkalinity is maintained at 0.6 to 0.7 lb. of free calcium oxide per ton of tailing water, as determined by titration with standard sulphuric acid solution, using phenolphthalein as an indicator. The *pH* value of the tailing water is from 12.0 to 13.0 or approaching the *pH* of this saturated lime water. A *pH* of this degree can be maintained with a titration of from 0.4 to 0.5, but our experience has been that more uniform results followed the higher titration.

3. *Minimum Amounts of Xanthate*—Sodium ethyl xanthate is a powerful and rapid collector, and it is essential that the consumption rate be maintained at 0.07 to 0.12 lb. per ton of flotation feed at the mill proper and at 0.20 to 0.25 lb. per ton at the slime plant.

4. *Frothing and Froth Modifying Oils*—Over-frothing in alkaline circuits usually results in a high tailing, and this statement is particularly true with regard to Anaconda ores.

5. *Uniform Tonnage Rate*—This factor is of utmost importance, as it has a direct bearing on all the preceding factors. With a fixed grinding circuit it is essential that full tonnage be secured by having at all times the maximum load that can be successfully ground, instead of a varying load that is but partly ground for some part of the time and overground the remainder of the time. Alkalinity and reagent control are both more easily maintained under a uniform tonnage rate.

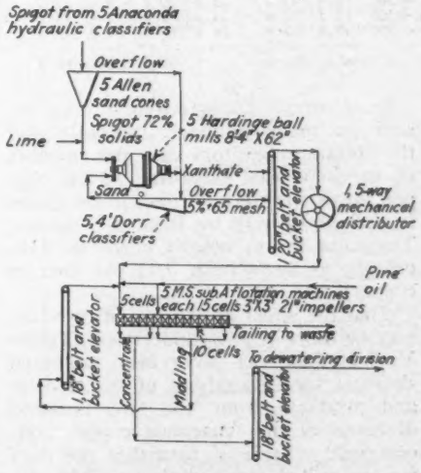



**Fig. 3—Grinding and flotation division, sections 2 to 7 inclusive, one section shown. Capacity each section, 1,500 tons.**

## Table V—Screen Analyses—Feed and Products Mill Flotation Division

| Mesh | Solids, Per Cent Held On | Solids, Per Cent Held Cum. | Copper Assay On | Copper Assay Cum. | Copper Per Cent Held On | Copper Per Cent Held Cum. | Iron (Fe) Assay On | Iron (Fe) Assay Cum. | Iron (Fe) Per Cent Held On | Iron (Fe) Per Cent Held Cum. | Sulphur Assay On | Sulphur Assay Cum. | Sulphur Per Cent Held On | Sulphur Per Cent Held Cum. | Silica ($SiO_2$) Assay On | Silica ($SiO_2$) Assay Cum. | Silica ($SiO_2$) Per Cent Held On | Silica ($SiO_2$) Per Cent Held Cum. | Alumina ($Al_2O_3$) Assay On | Alumina ($Al_2O_3$) Assay Cum. | Alumina ($Al_2O_3$) Per Cent Held On | Alumina ($Al_2O_3$) Per Cent Held Cum. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mill Flotation Feed** | | | | | | | | | | | | | | | | | | | | | | |
| + 48 | 0.4 | 0.4 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| + 65 | 6.5 | 6.9 | 0.25 | 0.25 | 0.4 | 0.4 | 2.1 | 2.1 | 1.2 | 1.2 | 0.7 | 0.7 | 0.3 | 0.3 | 85.4 | 85.4 | 10.3 | 10.3 | 7.6 | 7.6 | 7.6 | 7.6 |
| +100 | 11.2 | 18.1 | 0.49 | 0.37 | 1.4 | 1.8 | 1.8 | 1.9 | 1.7 | 2.9 | 1.8 | 1.4 | 1.4 | 1.7 | 84.1 | 84.6 | 16.4 | 26.7 | 7.5 | 7.6 | 12.1 | 19.7 |
| +150 | 9.3 | 27.4 | 1.49 | 0.77 | 3.5 | 5.3 | 5.7 | 3.2 | 4.5 | 7.4 | 6.9 | 3.3 | 4.4 | 6.1 | 76.4 | 81.8 | 12.5 | 39.2 | 6.6 | 7.2 | 8.9 | 28.6 |
| +200 | 15.1 | 42.5 | 3.26 | 1.65 | 12.4 | 17.7 | 14.4 | 7.2 | 18.5 | 25.9 | 18.4 | 8.6 | 18.9 | 25.0 | 52.6 | 71.4 | 13.9 | 53.1 | 4.6 | 6.3 | 10.0 | 38.6 |
| +300 | 6.9 | 49.4 | 5.17 | 2.14 | 9.0 | 26.7 | 17.3 | 8.6 | 10.1 | 36.0 | 19.6 | 10.2 | 9.1 | 34.1 | 46.9 | 68.0 | 5.7 | 58.8 | 4.4 | 6.0 | 4.4 | 43.0 |
| —300 | 50.6 | 50.6 | 5.76 | 5.76 | 73.3 | 73.3 | 14.9 | 14.9 | 64.0 | 64.0 | 19.2 | 19.2 | 65.9 | 65.9 | 46.6 | 46.6 | 41.2 | 41.2 | 7.8 | 7.8 | 57.0 | 57.0 |
| Totals | | 100.0 | | 3.97 | | 100.0 | | 11.8 | | 100.0 | | 14.7 | | 100.0 | | 57.2 | | 100.0 | | 6.9 | | 100.0 |
| **Mill Flotation Concentrate** | | | | | | | | | | | | | | | | | | | | | | |
| + 48 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| + 65 | 0.2 | 0.2 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| +100 | 0.3 | 0.5 | 18.7 | 18.7 | 0.3 | 0.3 | 14.2 | 14.2 | 0.3 | 0.3 | 25.6 | 25.6 | 0.3 | 0.3 | 29.8 | 29.8 | 2.1 | 2.1 | 2.4 | 2.4 | 0.6 | 0.6 |
| +150 | 4.5 | 5.0 | 22.9 | 22.5 | 3.5 | 3.8 | 22.4 | 21.6 | 5.0 | 5.3 | 35.6 | 34.7 | 4.8 | 5.1 | 11.0 | 12.8 | 7.4 | 9.5 | 1.0 | 1.1 | 2.6 | 3.2 |
| +200 | 12.2 | 17.2 | 24.7 | 24.1 | 10.1 | 13.9 | 25.6 | 24.5 | 15.4 | 20.7 | 39.6 | 38.2 | 14.3 | 19.4 | 3.5 | 6.2 | 6.3 | 15.8 | 0.7 | 0.8 | 4.9 | 8.1 |
| +300 | 26.4 | 43.6 | 28.0 | 26.4 | 25.0 | 38.9 | 24.7 | 24.6 | 32.3 | 53.0 | 38.9 | 38.6 | 30.4 | 59.8 | 2.0 | 3.7 | 7.9 | 23.7 | 0.5 | 0.6 | 7.6 | 15.7 |
| —300 | 56.4 | 56.4 | 32.1 | 32.1 | 61.1 | 61.1 | 16.8 | 16.8 | 47.0 | 47.0 | 30.0 | 30.0 | 50.2 | 50.2 | 9.1 | 9.1 | 76.3 | 76.3 | 2.6 | 2.6 | 84.3 | 84.3 |
| Totals | | 100.0 | | 29.6 | | 100.0 | | 20.2 | | 100.0 | | 33.8 | | 100.0 | | 6.7 | | 100.0 | | 1.7 | | 100.0 |
| **Mill Flotation Tailing** | | | | | | | | | | | | | | | | | | | | | | |
| + 48 | 0.4 | 0.4 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| + 65 | 7.8 | 8.2 | 0.19 | 0.190 | 6.4 | 6.4 | 1.8 | 1.8 | 1.4 | 1.4 | 0.7 | 0.7 | 0.5 | 0.5 | 87.5 | 87.5 | 10.8 | 10.8 | 6.0 | 6.0 | 6.5 | 6.5 |
| +100 | 10.9 | 19.1 | 0.16 | 0.173 | 7.3 | 13.7 | 1.6 | 1.7 | 1.6 | 3.0 | 1.6 | 1.2 | 1.5 | 2.0 | 87.2 | 87.2 | 14.4 | 25.2 | 5.5 | 5.7 | 8.0 | 14.5 |
| +150 | 9.8 | 28.9 | 0.18 | 0.176 | 7.3 | 21.0 | 5.4 | 2.9 | 4.8 | 7.8 | 6.0 | 2.8 | 4.9 | 6.9 | 80.1 | 84.9 | 11.8 | 37.0 | 6.7 | 6.0 | 8.7 | 23.2 |
| +200 | 13.7 | 42.6 | 0.18 | 0.177 | 10.2 | 31.2 | 13.1 | 6.2 | 16.6 | 24.4 | 14.7 | 6.7 | 17.0 | 23.9 | 67.8 | 79.4 | 14.1 | 51.1 | 5.3 | 5.8 | 9.7 | 32.9 |
| +300 | 9.1 | 51.7 | 0.23 | 0.186 | 8.7 | 39.9 | 17.0 | 8.1 | 14.2 | 38.6 | 18.5 | 8.7 | 14.1 | 38.0 | 58.5 | 75.7 | 8.0 | 59.1 | 4.5 | 5.6 | 5.4 | 38.3 |
| —300 | 48.3 | 48.3 | 0.30 | 0.300 | 60.1 | 60.1 | 13.8 | 13.8 | 61.4 | 61.4 | 15.3 | 15.3 | 62.0 | 62.0 | 56.0 | 56.4 | 40.8 | 40.9 | 9.6 | 9.6 | 61.7 | 61.7 |
| Totals | | 100.0 | | 0.241 | | 100.0 | | 10.9 | | 100.0 | | 11.9 | | 100.0 | | 66.2 | | 100.0 | | 7.5 | | 100.0 |

## TABLE VI—Screen Analyses—Feed and Products Slime Flotation Plant

| Mesh | Solids, Per Cent Held On | Solids, Per Cent Held Cum. | Copper Assay On | Copper Assay Cum. | Copper Per Cent Held On | Copper Per Cent Held Cum. | Iron (Fe) Assay On | Iron (Fe) Assay Cum. | Iron (Fe) Per Cent Held On | Iron (Fe) Per Cent Held Cum. | Sulphur Assay On | Sulphur Assay Cum. | Sulphur Per Cent Held On | Sulphur Per Cent Held Cum. | Silica ($Si_2$) Assay On | Silica ($Si_2$) Assay Cum. | Silica ($Si_2$) Per Cent Held On | Silica ($Si_2$) Per Cent Held Cum. | Alumina ($Al_2O_3$) Assay On | Alumina ($Al_2O_3$) Assay Cum. | Alumina ($Al_2O_3$) Per Cent Held On | Alumina ($Al_2O_3$) Per Cent Held Cum. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Flotation Feed—Slime Plant** | | | | | | | | | | | | | | | | | | | | | | |
| +150 | 0.7 | 0.7 | 1.11 | 1.11 | 0.2 | 0.2 | 2.7 | 2.7 | 0.3 | 0.3 | 2.9 | 2.9 | 0.2 | 0.2 | 74.8 | 74.8 | 0.9 | 0.9 | 9.5 | 9.5 | 0.4 | 0.4 |
| +200 | 2.4 | 3.1 | 0.75 | 0.83 | 0.4 | 0.6 | 1.6 | 1.8 | 0.7 | 1.0 | 2.0 | 2.2 | 0.6 | 0.8 | 79.4 | 78.4 | 3.3 | 4.2 | 9.8 | 9.7 | 1.4 | 1.8 |
| +300 | 5.9 | 9.0 | 1.72 | 1.42 | 2.4 | 3.0 | 3.0 | 2.6 | 3.0 | 4.0 | 3.8 | 3.3 | 2.8 | 3.6 | 75.3 | 76.3 | 7.9 | 12.1 | 9.2 | 9.4 | 3.4 | 5.2 |
| —300 | 91.0 | 91.0 | 4.43 | 4.43 | 97.0 | 97.0 | 6.1 | 6.1 | 96.0 | 96.0 | 8.4 | 8.4 | 96.4 | 96.4 | 54.4 | 54.4 | 87.9 | 87.9 | 16.9 | 16.9 | 94.8 | 94.8 |
| Total.... | 100.0 | | | 4.16 | | 100.0 | | 5.8 | | 100.0 | | 7.9 | | 100.0 | | 56.4 | | 100.0 | | 16.2 | | 100.0 |
| **Flotation Concentration—Slime Plant** | | | | | | | | | | | | | | | | | | | | | | |
| +150 | 0.3 | 0.3 | 16.1 | 16.1 | 0.3 | 0.3 | 20.8 | 20.8 | 0.4 | 0.4 | 31.6 | 31.6 | 0.4 | 0.4 | 21.1 | 21.1 | 0.5 | 0.5 | 1.7 | 1.7 | 0.1 | 0.1 |
| +200 | 0.8 | 1.1 | 17.8 | 17.3 | 0.6 | 0.9 | 23.7 | 22.8 | 0.9 | 1.3 | 34.1 | 33.3 | 0.8 | 1.2 | 14.8 | 16.8 | 0.7 | 1.2 | 1.4 | 1.5 | 0.2 | 0.3 |
| +300 | 2.0 | 3.1 | 20.4 | 19.3 | 2.0 | 2.9 | 24.6 | 23.9 | 2.4 | 3.7 | 37.0 | 35.7 | 2.3 | 3.5 | 9.6 | 12.2 | 1.3 | 2.5 | 1.0 | 1.2 | 0.4 | 0.7 |
| —300 | 96.9 | 96.9 | 20.3 | 20.3 | 97.1 | 97.1 | 19.6 | 19.6 | 96.3 | 96.3 | 31.4 | 31.4 | 96.5 | 96.5 | 15.0 | 15.0 | 97.5 | 97.5 | 5.4 | 5.4 | 99.3 | 99.3 |
| Total.... | 100.0 | | | 20.3 | | 100.0 | | 19.7 | | 100.0 | | 31.5 | | 100.0 | | 14.9 | | 100.0 | | 5.3 | | 100.0 |
| **Flotation Tailing—Slime Plant** | | | | | | | | | | | | | | | | | | | | | | |
| +150 | 0.9 | 0.9 | 0.30 | 0.30 | 0.7 | 0.7 | 1.6 | 1.6 | 0.7 | 0.7 | 1.0 | 1.0 | 0.4 | 0.4 | 79.4 | 79.4 | 1.1 | 1.1 | 8.7 | 8.7 | 0.5 | 0.5 |
| +200 | 3.7 | 4.6 | 0.16 | 0.19 | 1.4 | 2.1 | 0.3 | 0.6 | 0.5 | 1.2 | 0.9 | 0.9 | 1.4 | 1.8 | 81.8 | 81.3 | 4.5 | 5.6 | 9.5 | 9.3 | 2.1 | 2.6 |
| +300 | 6.9 | 11.5 | 0.17 | 0.18 | 2.9 | 5.0 | 0.3 | 0.4 | 0.9 | 2.1 | 1.2 | 1.1 | 3.4 | 5.2 | 81.0 | 81.1 | 8.4 | 14.0 | 9.9 | 9.7 | 4.1 | 6.7 |
| —300 | 88.5 | 88.5 | 0.44 | 0.44 | 95.0 | 95.0 | 2.4 | 2.4 | 97.9 | 97.9 | 2.6 | 2.6 | 94.8 | 94.8 | 64.6 | 64.6 | 86.0 | 86.0 | 17.4 | 17.4 | 93.3 | 93.3 |
| Total.... | 100.0 | | | 0.40 | | 100.0 | | 2.2 | | 100.0 | | 2.4 | | 100.0 | | 66.5 | | 100.0 | | 16.5 | | 100.0 |

## TABLE VII—Screen Analyses—Total Flotation Concentrate

| Mesh | Solids, Per Cent Held On | Solids, Per Cent Held Cum. | Copper Assay On | Copper Assay Cum. | Copper Per Cent Held On | Copper Per Cent Held Cum. | Iron (Fe) Assay On | Iron (Fe) Assay Cum. | Iron (Fe) Per Cent Held On | Iron (Fe) Per Cent Held Cum. | Sulphur Assay On | Sulphur Assay Cum. | Sulphur Per Cent Held On | Sulphur Per Cent Held Cum. | Silica ($Si_2$) Assay On | Silica ($Si_2$) Assay Cum. | Silica ($Si_2$) Per Cent Held On | Silica ($Si_2$) Per Cent Held Cum. | Alumina ($Al_2O_3$) Assay On | Alumina ($Al_2O_3$) Assay Cum. | Alumina ($Al_2O_3$) Per Cent Held On | Alumina ($Al_2O_3$) Per Cent Held Cum. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Total Flotation Concentrate** | | | | | | | | | | | | | | | | | | | | | | |
| +100 | 1.8 | 1.8 | 16.8 | 16.8 | 1.2 | 1.2 | 19.0 | 19.0 | 1.5 | 1.5 | 30.7 | 30.7 | 1.5 | 1.5 | 23.7 | 23.7 | 5.4 | 5.4 | 1.7 | 1.7 | 1.7 | 1.7 |
| +150 | 5.1 | 6.9 | 19.3 | 18.7 | 3.9 | 5.1 | 25.6 | 23.8 | 6.0 | 7.5 | 38.8 | 36.7 | 5.7 | 7.2 | 8.4 | 12.3 | 5.6 | 11.0 | 0.9 | 1.1 | 2.6 | 4.3 |
| +200 | 10.4 | 17.3 | 22.0 | 20.7 | 9.1 | 14.2 | 27.6 | 26.1 | 13.1 | 20.6 | 40.5 | 39.0 | 12.1 | 19.3 | 3.8 | 7.2 | 5.1 | 16.1 | 0.5 | 0.7 | 3.0 | 7.3 |
| +300 | 13.7 | 31.0 | 23.6 | 22.0 | 12.8 | 27.0 | 27.5 | 26.7 | 17.2 | 37.8 | 40.2 | 39.5 | 15.7 | 35.0 | 3.0 | 5.3 | 5.3 | 21.4 | 0.4 | 0.6 | 3.1 | 10.4 |
| —300 | 69.0 | 69.0 | 26.8 | 26.8 | 73.0 | 73.0 | 19.8 | 19.8 | 62.2 | 62.2 | 33.0 | 33.0 | 65.0 | 65.0 | 8.8 | 8.8 | 78.6 | 78.6 | 2.3 | 2.3 | 89.6 | 89.6 |
| Total.... | 100.0 | | | 25.3 | | 100.0 | | 22.0 | | 100.0 | | 35.0 | | 100.0 | | 7.7 | | 100.0 | | 1.8 | | 100.0 |

6. *Control*—Colorimetric tailing assays are made at hourly intervals, and the flotation operators vary the amounts of xanthate used according to the copper present therein. Concentrate grade is judged entirely by froth appearance. The color assays usually check the laboratory assays within 0.03 per cent of copper.

That the effect of fineness of grinding may be more readily understood, Tables V, VI, and VII have been prepared showing screen analyses of the feed to, and products from, the two flotation divisions of the Anaconda copper concentrator. Table V furnishes the data for the mill flotation division, Table VI for the slime division, and Table VII the combined mill and slime flotation concentrate.

To produce a total flotation concentrate of 25 per cent copper or better, two possibilities are open: First, to produce a concentrate from the deslimed ore assaying well in excess of 25 per cent copper and a slime concentrate of approximately 20 per cent copper; second, to produce a concentrate of approximately 25 per cent in each instance.

An examination of Table VI shows that the production of a concentrate assaying 25 per cent copper from Anaconda slime, is, to a large extent, a problem of eliminating finely divided silica and alumina. Our experience has been that forcing up the grade of concentrate from Anaconda slime always entails undue losses of copper, silver, and gold. Therefore, the logical place to work for high grade is on that portion of the ore that must be ground before treatment.

Table V shows that a copper content of 25 per cent is not reached in the concentrate except in that portion that is minus 200 mesh. This table also shows that grinding to 7 per cent plus 65 mesh produces a ground product containing but 18 per cent of the copper in particles coarser than 200 mesh, and that 73 per cent of the copper lies in the minus-300-mesh material. This is an illustration of the so-called selective grinding of the sulphides and is no doubt attributable to their relative hardness compared to the gangue, and to some extent, at least, to the disproportionate return of sulphide and gangue in the Dorr classifier circuit.

Table VI also illustrates the effect of overgrinding on copper losses in the

tailing—that is, the minus-300-mesh material assays 25 per cent higher than the total tailing. It is probably true that the extremely fine material—the equivalent of minus-1,000-mesh—would assay much higher in copper than the minus-300-mesh material, indicating that future improvements in recovery will depend, to a large extent, on the perfection of grinding circuits, wherein the desired fineness on the coarser sizes is maintained with a minimum of extremely fine materials.

The quantity of flotation agents required is quite low and is typified in Table VIII.

**TABLE VIII—Flotation Reagents Per Ton Flotation Feed**

| | At Mill | At Slime Plant | Per Ton of Ore Milled |
|---|---|---|---|
| Burned Lime | 5.7 | 5.8 | 5.6 |
| S. D. Pine Oil | 0.15 | 0.04 | 0.13 |
| Coal Tar Oil | 0.08 | .... | 0.06 |
| D. D. Pine Oil | .... | 0.04 | 0.01 |
| Xanthate | 0.10 | 0.25 | 0.13 |

Burned lime is made at the Anaconda plant, converted to milk of lime containing from 25 to 28 per cent solids, and then pumped to the various points of use. For that portion of the ore which is ground in ball mills, the lime is added directly to the mills. For the slime the lime is added to the pulp as it enters a set of conditioning machines, each consisting of a series of fifteen cells, each 3x3-ft. in cross-section and each equipped with a 24-in. cruciform impeller mounted on a vertical spindle. It is necessary to agitate Anaconda slime mechanically with lime to secure a uniform and consistently conditioned pulp. Xanthate is added to the slime pulp in the thirteenth conditioning cell and to the boots of the elevators feeding the flotation machines at the mill. The frothing oils are added at the head of the machines and the steam-distilled pine oil is added again about midway of each machine.

Production of a higher-grade concentrate at Anaconda is not so much a problem for the mill as for the smelter, as it is a matter of whether or not a higher-grade concentrate can be smelted as economically as a concentrate of 25 to 26 per cent copper. So far as the mill is concerned the production of a higher-grade concentrate is primarily a matter of proper grinding, and when smelter economies can balance the additional grinding cost, a higher grade concentrate can be made.

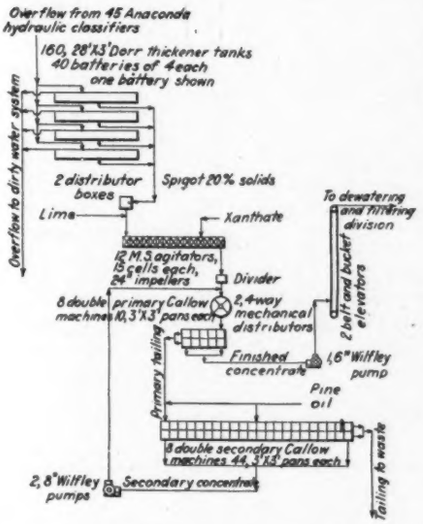

Overflow from 45 Anaconda hydraulic classifiers
160, 28'X3' Dorr thickener tanks 40 batteries of 4 each one battery shown
Overflow to dirty water system
2 distributor boxes
Spigot 20% solids
To dewatering and filtering division
Lime
Xanthate
12 M.S. agitators, 15 cells each, 24" impellers
Divider
8 double primary Callow machines
10, 3'X3' pans each
2, 4-way mechanical distributors
2 belt and bucket elevators
Finished concentrate
4, 6" Wilfley pump
Primary tailing
Pine oil
8 double secondary Callow machines 44, 3'X3' pans each
2, 8" Wilfley pumps
Secondary concentrate
Tailing to waste

**Fig. 4—Flow sheet of slime division of copper concentrator for all eight sections**

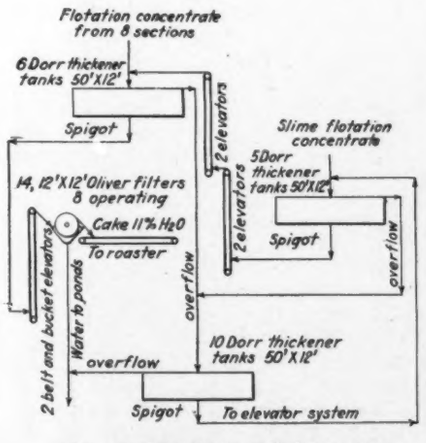



**Fig. 5—Dewatering division for eight sections**

THE ZINC ORES concentrated at Anaconda are sulphide ores from Butte, and contain less than 3 per cent iron as pyrite. The principal minerals present are sphalerite (ZnS), galena (PbS), and pyrite ($FeS_2$). Copper is present in small amounts, and silver and gold are associated with both the sphalerite and with the galena. A small amount of gold is associated with the pyrite. The gangue is quartz, highly altered granite, and feldspar, and in the ore from the Emma Mine, rhodochrosite or manganese carbonate.

A typical analysis of the present mill heads follows:

| | Per Cent | Oz. per Ton |
|---|---|---|
| Zinc | 12.7 | ..... |
| Lead | 2.2 | .... |
| Silver | .... | 6.4 |
| Gold | .... | 0.008 |
| Copper | 0.16 | ..... |
| Iron | 2.8 | ..... |
| Sulphur | 8.6 | ..... |
| Silica | 56.2 | ..... |
| Alumina | 5.0 | ..... |
| Lime | 1.4 | ..... |

The zinc concentrating plant was built in 1916 and was operated until 1924 as a collective flotation plant—that is, no attempt was made to separate one sulphide from another. The primary crushing is done in two stages with jaw crushers. Rolls in three stages, with the necessary trommels, reduce the ore to 1½ mm. It is then ground in Hardinge ball mills operating in closed circuit with Dorr classifiers and floated.

The selective problem presented is twofold: the rejection of barren pyrite and the separation of sphalerite and galena. The scheme adopted is shown in detail by the accompanying flow sheets, Figs. 6, 7, and 8, and was originally chosen because it entailed the minimum changes in equipment and was applicable to an ore containing sphalerite as the principal, and galena as the

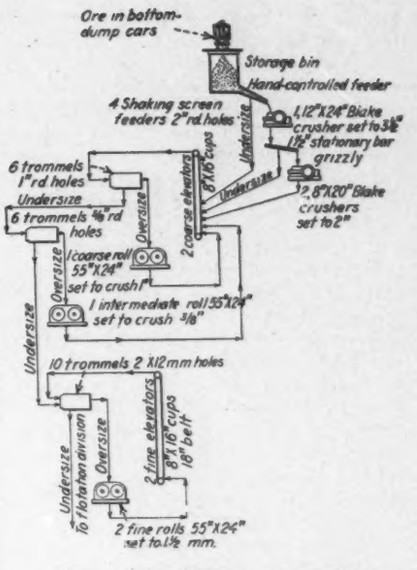

Ore in bottom-dump cars
Storage bin
Hand-controlled feeder
4 Shaking screen feeders 2" rd. holes
4, 12"X24" Blake crusher set to 3½"
1½" stationary bar grizzly
6 trommels 1" rd holes
Undersize
8'X16' cups
Oversize
2 coarse elevators
2, 8"X20" Blake crushers set to 2"
6 trommels ⅝" rd holes
1 coarse roll 55"X24" set to crush 1"
1 intermediate roll 55"X24" set to crush ⅜"
10 trommels 2 X12 mm holes
2 fine elevators 8"X16" cups 18" belt
Undersize to flotation division
Oversize
2 fine rolls 55"X24" set to 1½ mm.

**Fig. 6—Flow sheet of crushing division, zinc concentrator**

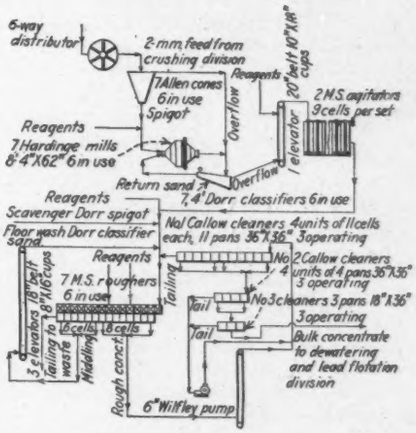

6-way distributor
2-mm. feed from crushing division
7 Allen cones 6 in use
Spigot
Reagents
Overflow
20" belt 10"X16" cups
2 M.S. agitators 9 cells per set
7 Hardinge mills 8'4"X62" 6 in use
1 elevator
Return sand
7, 4' Dorr classifiers 6 in use
Scavenger Dorr spigot
Floor wash Dorr classifier
No. 1 Callow cleaners 4 units of 11 cells each, 11 pans 36"X36" 3 operating
7 M.S. roughers 6 in use
No. 2 Callow cleaners 4 units of 4 pans 36"X36" 3 operating
No. 3 cleaners 3 pans 18"X36" 3 operating
Tail
Middling
Rough concl.
Bulk concentrate to dewatering and lead flotation division
6" Wilfley pump

**Fig. 7—Grinding and bulk flotation division, zinc concentrator**

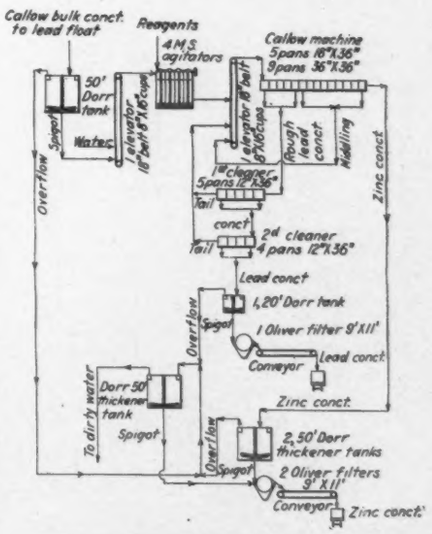



**Fig. 8—Dewatering and lead float division, zinc concentrator**

**Return of spigot from 10 overflow thickener tanks to 5 slime thickener tanks and 6 mill thickener tanks is made to mix slime and granular concentrate and thereby facilitate thickener operation.**

accessory, valuable sulphide. In brief, this scheme is first to float the sphalerite and galena in the form of a bulk or collective concentrate containing a minimum amount of pyrite and gangue, and subsequently to refloat this concentrate for the production of a high-grade lead concentrate and a lead tailing which is a relatively high-grade zinc concentrate.

Inasmuch as the zinc concentrator is one division of the Anaconda electrolytic plants, the mark set was a high recovery of zinc in the zinc concentrate, even though such procedure lowered the zinc assay of this product and the recovery of lead in the lead concentrate, owing to dilution of the zinc concentrate with galena. The selective practices followed fall into two main divisions, the flow sheets used being virtually the same, though the reagent combinations used for effecting the bulk or collective float are decidedly different. In general, the flow sheet followed in each case is the production of a rough concentrate in standard M. S. machines, followed by triple cleaning in Callow cells for the production of the bulk concentrate. The bulk concentrate is then thickened to 50 per cent solids to eliminate, so far as possible, the effect of the agents used for activating sphalerite and for producing froth. The thickened concentrate is then mechanically agitated in a series of agitators to which a sphalerite depressant has been added, diluted with fresh water, and the galena floated.

Reagent combinations used from 1924 to about January, 1927, were, for the bulk float: Burned lime, used as milk of lime, for alkalinity; thio-carbanilid dissolved in cresylic acid and orthotoluidine as the principal collector, copper sulphate as the activating agent for sphalerite; pine oil for froth production.

For the lead-zinc separation: Zinc sulphate and sodium cyanide as the sphalerite depressant.

Except in rare instances no frothing agents are required for the lead float.

Since January, 1929, the reagents used have been, for the bulk float: Burned lime in the form of milk of lime for alkalinity; sodium ethyl xanthate as the collector; copper sulphate for activation of sphalerite; less than 0.05 lb. of calcium cyanide for depression of pyrite; pine oil as the primary frother, and a coal tar oil as a froth stabilizer.

For the lead-zinc separation, zinc sulphate and calcium cyanide are used instead of zinc sulphate and sodium cyanide. In each case pyrite rejection is easily accomplished.

The reagent combinations recently used are more economical and have resulted in no material change in grade of concentrates or in the metallic content of tailings. Typical metallurgical results with the former and present reagents are shown in Tables IX and X.

The technique depends upon:

Feed rate control; proper alkalinity at all times; fineness of grinding; machine regulation; control of volume and per cent of solids in the thickened bulk concentrate to lead flotation; proper control of flotation agents.

Each of the foregoing factors is of prime importance and they are all so interlocked that it is difficult to say which is of the most importance.

Owing to the lead-zinc separation, the reagent costs are high, but when the nature of the problem is considered they are not excessive.

**TABLE IX—Assays, Using Thio-carbanilid, Cresylic Acid, and Orthotoluidine**

| | Per Cent | | | Oz. per Ton | | Per Cent | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Zn | Pb | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
| Mill feed | 12.2 | 2.1 | 0.22 | 5.2 | 0.007 | 3.0 | 8.4 | 56.2 | 5.4 | 0.8 |
| Zinc concentrate | 53.1 | 5.0 | 0.68 | 18.5 | 0.023 | 3.0 | 28.7 | 5.1 | 1.0 | 0.7 |
| Lead concentrate | 6.0 | 71.8 | 2.59 | 44.6 | 0.017 | 1.5 | 18.1 | 0.2 | 0.3 | 0.7 |
| Tailing | 0.8 | 0.2 | 0.07 | 0.7 | 0.004 | 3.0 | 71.3 | 2.6 | 6.7 | 1.0 |

**TABLE X—Assays, Using Sodium Ethyl Xanthate**

| | Per Cent | | | Oz. per Ton | | Per Cent | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Zn | Pb | Cu | Ag | Au | Fe | S | $SiO_2$ | $Al_2O_3$ | CaO |
| Mill feed | 12.3 | 2.3 | 0.19 | 5.8 | 0.009 | 3.4 | 8.6 | 56.3 | 4.3 | 1.4 |
| Zinc concentrate | 53.1 | 5.3 | 0.54 | 20.1 | 0.026 | 4.0 | 29.7 | 3.8 | 0.3 | 0.5 |
| Lead concentrate | 6.1 | 69.7 | 3.0 | 58.9 | 0.020 | 1.5 | 16.2 | 0.1 | 0.1 | 0.4 |
| Tailing | 0.8 | 0.25 | 0.07 | 0.8 | 0.003 | 2.7 | 2.4 | 71.9 | 6.4 | 1.5 |

**Reagent Consumption Per Ton of Ore**

| | Pounds |
|---|---|
| Burned lime | 3.50 |
| Sodium ethyl xanthate | 0.20 |
| Steam-distilled pine oil | 0.17 |
| Coal tar oil | 0.08 |
| Calcium cyanide | 0.24 |
| Zinc sulphate | (a) 2.27 |
| Copper sulphate | 1.69 |

(a) 2.27 lb. of solution containing 10 per cent zinc.

Screen analyses of zinc concentrator flotation feed and products are appended (Table XI), so that the grinding requirements may be more readily understood. Fine grinding is necessary for both recovery and grade. Over-grinding is not so fatal as at the copper concentrator, but is not especially desirable.

I am particularly indebted to Mr. S. S. Rodgers, superintendent of the copper concentrator; Mr. C. W. Morse, superintendent of the zinc concentrator; and Mr. W. H. Casto, assistant to Mr. Rodgers, for their co-operation in solving concentrating problems and in plant operation, and to Mr. F. F. Frick, research engineer, and his staff, for their co-operation in the working out of Anaconda's selective flotation practice.

**Table XI—Zinc Concentrator Flotation Feed and Products Screen Analyses**

| Mesh | Solids Per Cent Held On | Solids Per Cent Held Cum. | Zinc Assay On | Zinc Assay Cum. | Zinc Per Cent Held On | Zinc Per Cent Held Cum. | Lead Assay On | Lead Assay Cum. | Lead Per Cent Held On | Lead Per Cent Held Cum. | Silver Assay On | Silver Assay Cum. | Silver Per Cent Held On | Silver Per Cent Held Cum. | Iron Assay On | Iron Assay Cum. | Iron Per Cent Held On | Iron Per Cent Held Cum. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Flotation Feed** | | | | | | | | | | | | | | | | | | |
| + 48 | 1.0 | 1.0 | 5.2 | 5.2 | 0.4 | 0.4 | 0.35 | 0.35 | 0.2 | 0.2 | 2.4 | 2.4 | 0.4 | 0.4 | 1.9 | 1.9 | 0.9 | 0.9 |
| + 65 | 5.2 | 6.2 | 6.7 | 6.5 | 2.4 | 2.8 | 0.45 | 0.43 | 1.3 | 1.5 | 2.8 | 2.7 | 2.3 | 2.7 | 1.6 | 1.7 | 3.9 | 4.8 |
| +100 | 7.5 | 13.7 | 10.2 | 8.5 | 5.3 | 8.1 | 0.70 | 0.58 | 3.0 | 4.5 | 3.9 | 3.4 | 4.6 | 7.3 | 1.6 | 1.6 | 5.6 | 10.4 |
| +150 | 8.1 | 21.8 | 12.6 | 10.0 | 7.0 | 15.1 | 0.85 | 0.68 | 3.9 | 8.4 | 4.7 | 3.9 | 5.9 | 13.2 | 1.9 | 1.7 | 7.2 | 17.6 |
| +200 | 11.3 | 33.1 | 15.9 | 12.0 | 12.5 | 27.6 | 1.20 | 0.86 | 7.7 | 16.1 | 5.5 | 4.4 | 9.8 | 23.0 | 3.0 | 2.2 | 15.9 | 33.5 |
| +300 | 17.4 | 50.5 | 16.0 | 13.4 | 19.3 | 46.9 | 1.85 | 1.20 | 18.2 | 34.3 | 6.6 | 5.2 | 18.0 | 41.0 | 2.2 | 2.2 | 17.9 | 51.4 |
| —300 | 49.5 | 49.5 | 15.5 | 15.5 | 53.1 | 53.1 | 2.35 | 2.35 | 65.7 | 65.7 | 7.6 | 7.6 | 59.0 | 59.0 | 2.1 | 2.1 | 48.6 | 48.6 |
| Totals | | 100.0 | | 14.4 | | 100.0 | | 1.77 | | 100.0 | | 6.4 | | 100.0 | | 2.1 | | 100.0 |
| **Zinc Concentrate** | | | | | | | | | | | | | | | | | | |
| + 65 | 1.2 | 1.2 | .... | .... | .... | .... | .... | .... | .... | .... | 14.2 | 14.2 | 4.3 | 4.3 | .... | .... | .... | .... |
| +100 | 4.4 | 5.6 | 52.2 | 52.2 | 5.3 | 5.3 | 2.0 | 2.0 | 3.4 | 3.4 | 14.2 | 14.2 | 4.3 | 4.3 | 2.3 | 2.3 | 4.2 | 4.2 |
| +150 | 7.8 | 13.4 | 52.8 | 52.6 | 7.5 | 12.8 | 2.3 | 2.2 | 5.4 | 8.8 | 14.4 | 14.4 | 6.2 | 10.5 | 3.3 | 2.9 | 8.4 | 12.6 |
| +200 | 10.5 | 23.9 | 53.5 | 53.0 | 10.2 | 23.0 | 2.7 | 2.4 | 8.6 | 17.4 | 14.8 | 14.8 | 8.7 | 19.2 | 3.7 | 3.2 | 12.7 | 25.3 |
| +300 | 20.0 | 43.9 | 55.3 | 54.0 | 20.1 | 43.1 | 2.2 | 2.3 | 13.3 | 30.7 | 15.1 | 15.1 | 16.9 | 36.1 | 3.9 | 3.5 | 25.4 | 50.7 |
| —300 | 56.1 | 56.1 | 56.0 | 56.0 | 56.9 | 56.9 | 4.1 | 4.1 | 69.0 | 69.3 | 21.0 | 21.0 | 63.9 | 63.9 | 3.7 | 2.7 | 49.3 | 49.3 |
| Totals | | 100.0 | | 55.1 | | 100.0 | | 3.3 | | 100.0 | | 18.4 | | 100.0 | | 3.1 | | 100.0 |
| **Lead Concentrate** | | | | | | | | | | | | | | | | | | |
| +150 | 1.56 | 1.56 | 9.9 | 9.9 | 2.1 | 2.1 | 38.0 | 38.0 | 0.9 | 0.9 | 188.6 | 188.6 | 3.3 | 3.3 | 5.0 | 5.0 | 3.3 | 3.3 |
| +200 | 5.11 | 6.67 | 8.9 | 9.1 | 6.4 | 8.5 | 60.0 | 54.8 | 4.5 | 5.4 | 115.2 | 132.4 | 6.6 | 9.9 | 3.5 | 3.9 | 7.6 | 10.9 |
| +300 | 25.59 | 32.26 | 8.9 | 9.0 | 31.5 | 40.0 | 67.1 | 64.6 | 25.1 | 30.5 | 81.6 | 92.1 | 23.4 | 33.3 | 2.9 | 3.1 | 31.5 | 42.4 |
| —300 | 67.74 | 67.74 | 6.4 | 6.4 | 60.0 | 60.0 | 70.0 | 70.0 | 69.5 | 69.5 | 87.9 | 87.9 | 66.7 | 66.7 | 2.0 | 2.0 | 57.6 | 57.6 |
| Totals | | 100.0 | | 7.2 | | 100.0 | | 68.2 | | 100.0 | | 89.3 | | 100.0 | | 2.4 | | 100.0 |
| **Zinc Tailing** | | | | | | | | | | | | | | | | | | |
| + 48 | 1.1 | 1.1 | 3.05 | 3.05 | 3.8 | 3.8 | 0.30 | 0.30 | 1.2 | 1.2 | 2.0 | 2.0 | 2.4 | 2.4 | 2.2 | 2.2 | 1.4 | 1.4 |
| + 65 | 5.4 | 6.5 | 1.90 | 2.09 | 11.7 | 15.5 | 0.25 | 0.26 | 5.0 | 6.2 | 1.4 | 1.5 | 8.3 | 10.7 | 1.2 | 1.4 | 3.9 | 5.3 |
| +100 | 8.0 | 14.5 | 1.00 | 1.49 | 9.0 | 24.5 | 0.18 | 0.22 | 5.6 | 11.8 | 1.0 | 1.2 | 8.7 | 19.4 | 1.1 | 1.2 | 5.2 | 10.5 |
| +150 | 9.6 | 24.1 | 0.80 | 1.22 | 8.7 | 33.2 | 0.17 | 0.20 | 6.1 | 17.9 | 0.8 | 1.1 | 8.4 | 27.8 | 1.2 | 1.2 | 6.9 | 17.4 |
| +200 | 12.0 | 36.1 | 0.55 | 0.99 | 7.4 | 40.6 | 0.20 | 0.20 | 8.9 | 26.8 | 0.7 | 0.9 | 9.2 | 27.0 | 1.7 | 1.4 | 12.1 | 29.5 |
| +300 | 16.0 | 52.1 | 0.40 | 0.81 | 7.3 | 47.9 | 0.18 | 0.19 | 10.8 | 37.6 | 0.6 | 0.8 | 10.6 | 47.6 | 1.7 | 1.5 | 16.3 | 45.8 |
| —300 | 47.9 | 47.9 | 0.95 | 0.95 | 52.1 | 52.1 | 0.35 | 0.35 | 62.4 | 62.4 | 1.0 | 1.0 | 52.4 | 52.4 | 1.9 | 1.9 | 54.2 | 54.2 |
| Totals | | 100.0 | | 0.88 | | 100.0 | | 0.27 | | 100.0 | | 0.9 | | 100.0 | | 1.7 | | 100.0 |

# Development of Copper Smelting at Anaconda

## *Present Practice at Plants in Montana, Arizona, and Utah Evolved From Half a Century's Experience*

By Louis V. Bender

*General Superintendent, Anaconda Reduction Works*

THE ORIGINAL smelting plant at Anaconda, Mont., known as the "Old Works," information concerning which was given to me by Captain William Kelly, general foreman, and Patrick Dalton, converter superintendent, was built in 1883 along the north side of Warm Springs Creek, just east of the present city limits of Anaconda. There were two main divisions, known as the Upper Works and the Lower Works. The smelter equipment consisted of roasters, reverberatory furnaces, and blast furnaces at each works, plus converters at the Lower Works. A fire occurred in 1889, destroying some of the equipment, but construction was begun at once, and replacement, as well as additional equipment, installed in both divisions. The Old Works was operated until the early part of 1902.

Construction of the New Works was begun in 1900 and finished in 1902, operations being started in the early part of that year. The New Plant was located on the south side of Warm Springs Creek, two miles east of the city of Anaconda. The smelting equipment consisted of roasters, a briquet plant, blast furnaces, reverberatory furnaces, converters, casting furnaces, and the usual additional equipment necessary for the continuity of operations, such as power houses, tramming system, and shops. With the exception of the briquet and the blast-furnace plants, these divisions, with some modifications, still exist.

At first there was only one roaster building, but in 1916, Roaster Building No. 2 was erected, the roasters therein being slightly different in design and also of greater hearth area than those in the No. 1 building. The briquet and blast-furnace plants were shut down in 1918 and the buildings and equipment scrapped in 1927 to make room for the electrolytic zinc plant.

When the blast furnaces were shut down it was necessary to find another means of treating the converter slag, which had hitherto been treated in the blast furnaces. It was not, at that time, considered feasible to treat this slag in the main reverberatories, because they were at a considerable distance from the converter building, and at a higher elevation. Consequently, a reverberatory furnace, known as the "converter slag furnace," was built in the converter building in the space formerly occupied by the lining department.

During the early operation of the New Works, the fumes and gases were discharged through stacks located at each of the main divisions. In 1903, flues from each of these divisions were led into a main flue, this leading to a stack at quite a distance from the furnaces and at a higher elevation. Soon afterward the individual stacks were taken down.

The first main stack was replaced with a larger one in 1918. Cottrell treaters were installed in the flues leading to the larger stack after an investigation of this equipment was made by installing an experimental Cottrell unit in the flue from Roaster No. 2. A reverberatory furnace was built near the main Cottrell plant to handle the dust recovered. In 1927, this furnace was shut down, and the Cottrell dust is now treated in modified McDougal furnaces in Roaster No. 1 building.

The Upper Works began operations in 1883 with 34 hand-rabbled reverberatory roasting furnaces 50 ft. long by 14 ft. wide, and 26 reverberatory matting furnaces, 20 ft. long by 14 ft. wide. The reverberatory matting furnaces smelted fifteen tons per day per furnace. This plant was remodeled in 1890, after which it remained unchanged until it was shut down in 1902. When shut down the plant had forty Bruckner roasting furnaces, 18 ft. long by 9 ft. in diameter. Each furnace roasted sixteen tons of charge per day, reducing the sulphur from 35 to 12 per cent, and using lignite coal in doing so. There were four Wethey roasters, 100 ft. long by 12 ft. wide, and eleven reverberatory smelting furnaces, with hearths 31 ft. long and 16 ft. wide, which treated 60 tons of charge per day per furnace, on the average.

The Lower Works, which was one mile distant from the Upper Works, was begun in 1887. Here were 56 Bruckner roasting furnaces, each 18 ft. long and 9 ft. in diameter. Each of the 28 reverberatory matting furnaces, 22 ft. long and 16 ft. wide, smelted 40 tons per day. The buildings were of wood and were burned down in 1889, but were rebuilt at once, replacement being made with steel buildings and the same equipment used. During 1890, 40 Bruckner roasters of the same size as the original ones were added to the plant, making a total of 96. In 1899, four McDougal furnaces of the Evans-Klepetko type were added, each furnace being 16 ft. in diameter and 21 ft. high, and being built with six hearths.

Seven of the original 28 matte furnaces were enlarged in 1899; one of them to a hearth of 32x18 ft.; one to 33x18 ft.; one to 45x18 ft., and four to hearths of 35x18 ft. They smelted an average of 75 tons per day per furnace.

Construction of the converter plant was begun in 1890; this had fifteen stalls for upright vessels, the converters being 10 ft. high and 6 ft. in diameter. In the converter building also were six water-jacketed cupola remelting furnaces, and seven blast furnaces, the latter being 8 ft. long by 42 in. wide at the tuyère level, and 10 ft. high.

When the New Works were being planned, it was considered better to build an entirely new plant instead of remodeling and enlarging the Old Works. A new location was chosen across Warm Springs Creek Valley, which site lent itself well, both from the standpoint of contour of ground and for extensions. The plant was arranged so that additions and alterations could be made with ease.

IN ROASTER No. 1, 48 roasters of the McDougal type, 16 ft. in diameter, were originally installed, each furnace having six hearths with a working floor at every other hearth. They were arranged in rows of four furnaces each, two rows or eight furnaces forming a battery. During 1905 two batteries were added, thus giving 64 furnaces in all. The arms were water-cooled. The furnaces had a capacity of 40 tons per day, roasting from 35 to 8 per cent of sulphur. Fumes and gases were carried away in sheet iron and brick flues to a dust chamber 300 ft. long, 40 ft. wide, and 40 ft. high.

During 1916, Roaster No. 2 build-

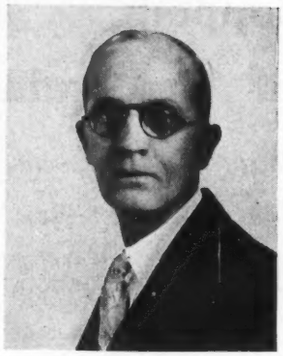
| Sulphur | | | | "Insoluble" | | | |
|---|---|---|---|---|---|---|---|
| Assay | | Per Cent | Held | Assay | | Per Cent | Held |
| On | Cum. | On | Cum. | On | Cum. | On | Cum. |
| 3.2 | 3.2 | 0.3 | 0.3 | 83.2 | 83.2 | 1.3 | 1.3 |
| 4.5 | 4.3 | 2.3 | 2.6 | 80.4 | 80.9 | 6.5 | 7.8 |
| 6.5 | 5.5 | 4.8 | 7.4 | 74.6 | 77.4 | 8.8 | 16.6 |
| 9.2 | 6.9 | 7.2 | 14.6 | 67.4 | 73.6 | 8.5 | 25.1 |
| 10.5 | 9.1 | 14.8 | 29.4 | 62.8 | 71.0 | 11.7 | 36.8 |
| 11.3 | 9.9 | 19.1 | 48.5 | 59.4 | 67.0 | 16.2 | 53.0 |
| 10.7 | 10.7 | 51.5 | 51.5 | 60.6 | 60.6 | 47.0 | 47.0 |
| | 10.3 | | 100.0 | | 63.8 | | 100.0 |
| 29.2 | 29.2 | 5.2 | 5.2 | 12.2 | 12.2 | 13.3 | 13.3 |
| 30.4 | 29.9 | 17.7 | 12.9 | 9.0 | 10.3 | 13.8 | 27.1 |
| 31.2 | 30.5 | 10.5 | 23.4 | 6.2 | 8.5 | 12.7 | 39.8 |
| 32.3 | 31.3 | 20.8 | 44.2 | 4.2 | 6.6 | 16.4 | 56.2 |
| 31.0 | 31.0 | 55.8 | 55.8 | 4.0 | 4.0 | 43.8 | 43.8 |
| | 31.1 | | 100.0 | | 5.1 | | 100.0 |
| | | | | Copper | | | |
| 22.0 | 22.0 | 2.0 | 2.0 | 8.61 | 8.61 | 3.9 | 3.9 |
| 19.2 | 19.9 | 5.7 | 7.7 | 5.42 | 6.17 | 8.1 | 12.0 |
| 17.2 | 17.7 | 25.8 | 33.5 | 3.20 | 3.81 | 24.0 | 36.0 |
| 16.8 | 16.8 | 66.5 | 66.5 | 3.22 | 3.22 | 64.0 | 64.0 |
| | 17.1 | | 100.0 | | 3.41 | | 100.0 |
| | | | | Insoluble | | | |
| 2.4 | 2.4 | 1.4 | 1.4 | 84.6 | 84.6 | 1.1 | 1.1 |
| 1.7 | 1.8 | 4.9 | 6.3 | 89.2 | 88.4 | 5.8 | 6.9 |
| 1.7 | 1.8 | 7.1 | 13.4 | 87.8 | 88.1 | 8.3 | 15.2 |
| 1.6 | 1.7 | 8.1 | 21.5 | 87.2 | 87.7 | 10.0 | 25.2 |
| 1.8 | 1.7 | 10.1 | 31.6 | 85.8 | 87.1 | 17.3 | 37.5 |
| 1.8 | 1.7 | 15.3 | 46.9 | 83.8 | 86.1 | 16.1 | 53.6 |
| 2.1 | 2.1 | 53.1 | 53.1 | 81.0 | 81.0 | 46.4 | 46.4 |
| | 1.9 | | 100.0 | | 83.6 | | 100.0 |

ing was constructed, and for several years both plants were used for roasting cuprous materials. At present all calcining is done in No. 2 building. The furnaces in No. 1 building have been changed somewhat and the building is now being used for different operations: arsenic roasting, roasting zinc plant residue, drying materials, and for storage space.

ROASTER No. 2 contains 23 furnaces of the Anaconda-McDougal-Wedge type, 25 ft. in diameter, with seven hearths. The arms are water-cooled. The top hearth is open and is used as a drying hearth. Furnace capacity varies with the character of the feed and with the amount of sulphur to be eliminated. Until a couple of years ago the tonnage was 40 tons of dry feed per day, roasting from 36 per cent sulphur to 8 per cent. This work was done on a feed consisting of table, jig, and flotation concentrates, with small quantities of fine ore, plant cleanings, and limerock. Under these conditions no fuel was required for actual roasting.

During 1927, the metallurgy in the concentrator was changed, whereby the copper assay of the concentrate was more than doubled, with a consequent cutting in two of the tonnage. Coincident with this an electrolytic zinc plant was erected at Anaconda. The 28 furnaces in Roaster No. 2 building were then divided equally between zinc and copper. The copper furnaces, with the new feed consisting of flotation concentrate, fine ore, limerock, and precipitates, now roast 200 tons of feed per day per furnace, bringing a 32 per cent sulphur feed down to an 18 per cent sulphur calcine. Little fuel is required in this operation, small quantities of oil being used whenever necessary.

THE briquet plant was operated in conjunction with the blast furnaces. Briquets were made from a mixture of slime, concentrate, fine ore, and jig coke, As first built the plant contained two Chisholm-Boyd and two White briquetting presses of a capacity of 125 tons each per 24 hours. The briquets were dried in an adjoining shed. Later, the original presses were replaced by four end-cut augur briquet machines made by Chambers Brothers Company, of Philadelphia. Each of these machines turned out an average of 840 tons per day. The briquets weighed 6 to 8 lb. each and were not dried, but conveyed to a row of hoppers placed directly over a track. Each hopper had a capacity of 1,000 lb., was steel-lined, and had a drop bottom through which the briquets were dropped into a charge car which had previously been partially loaded with weighed quantities of the various materials constituting a charge for the blast furnaces, and trammed directly to them. This plant was shut down in 1918 and the building and machinery were scrapped in 1927 to make room for the electrolytic zinc plant which now occupies the site.



**Louis V. Bender**

THE blast-furnace installation originally consisted of seven furnaces, each 15 ft. long by 56 in. wide at the tuyère level. They were built in a row, their longer axis being parallel to the length of the building, making charging from tracks on each side more convenient. In a short time No. 1 and No. 2 furnaces, which were spaced 21 ft. apart, were joined, thus making the reconstructed furnace 51 ft. long. A similar procedure was followed with Nos. 3 and 4 furnaces and with Nos. 5 and 6. The results were so satisfactory with the increase in length that it was decided to join No. 7 furnace with Nos. 5 and 6, making in this combination a single furnace 87 ft. long. Thus the original seven furnaces were made into three furnaces.

Each of the original furnaces had one settler, 16 ft. in diameter and 4 ft. high, water-cooled by spray pipes. These same settlers or forehearths served for the enlarged furnaces. Below the forehearth level was a track upon which the matte was conveyed in ladle cars to the converters. The slag ran off continually into a stream of water and was then sluiced in a launder to the dump. The matte assayed 46 to 50 per cent copper and the slag 0.25 to 0.30 per cent.

All blast-furnace equipment was scrapped in 1927 and the building remodeled for the use of the casting division of the electrolytic zinc plant, built in the following year.

REVERBERATORY furnaces were originally 50 ft. long and 19 ft. wide, but in time the length was increased, one furnace to 60 ft.; another to 85 ft.; another to 102 ft.; and another to 116 ft. These furnaces were heated from a firebox 7 ft. wide by 16 ft. long, the width being parallel to the length of the furnace. Coal in lump form was burned on a grate, the larger portion coming from the company's mines at Diamondville, Wyo., with a B.t.u. value of 11,500 to 12,000. The material smelted per day in the largest furnace averaged 250 tons, with a fuel ratio of 4 to 1. It was almost invariably true that as the furnaces were lengthened the copper assay of the slags decreased and the fuel ratio improved. The slags were granulated in water and sent by launder to the dump.

In 1914, one furnace was equipped for firing with pulverized coal. Results were so good that by 1916 all eight of the furnaces were changed to coal-dust firing. At the same time the furnaces were increased in size to hearth dimensions of 133 ft. long by 23½ ft. wide. Following these changes the tonnage smelted and the fuel ratio were both greatly increased. During 1928 and the early part of 1929, six of the furnaces were entirely surrounded with jackets with water pipes cast in them. These furnaces are now being operated as standard practice and are proving to be very satisfactory. Waste-heat boilers have always been used at Anaconda in connection with reverberatories, the steam being used to operate air compressors in the smelter power house.

COAL is pulverized in a plant that is close to the reverberatories. It has a capacity of 1,000 tons of coal per day ground to a fineness of 96 per cent through 100 mesh and 74 per cent through 200 mesh. The equipment consists of two 30x30 Jeffries single roll crushers; three Ruggles-Coles driers; twelve 4½-ft. Raymond pulverizers, and the necessary conveyors, elevators, bins, and various other complementary appurtenances.

The pulverized coal is taken by a screw conveyor to bins directly in front of the furnaces, from which it is fed to the furnaces through Warford burners. Primary air of 16-oz. pressure is used and in addition a variable quantity of secondary air is admitted through openings surrounding the burner pipes, the size of the openings being varied by means of slide dampers. In addition to supplying the reverberatory needs, pulverized coal is furnished to the converter slag furnace, to the casting furnaces, to the two rotary lime burners, and to other outside sources.

FOR converting, eight stalls were provided in the original plant. This number was later increased to eleven. There were fifteen converter shells of the barrel type, 8 ft. in diameter by 12 ft. 6 in. long and of eight tons' capacity. The shells were lined with siliceous ores, a lining that would last twelve hours. Silica for the slag was obtained from the lining.

The building was provided with two 60-ton electric cranes to take worn-out shells to the lining department and to bring newly lined shells back to the stalls; to convey the blister copper from the converters to the casting furnaces; to take the converter slag to the slag conveyors; and for other uses on the converter floor. Air for blowing was supplied at 15-lb. pressure, and the converters were turned over by hydraulic power of 300-lb. pressure.

By 1915, the eleven converter stalls were reduced to seven, and 20-ft. Great Falls type converters were installed, which is the equipment now in use. The converters are lined with magnesite brick, the silica for the slag being

provided by charges of siliceous ore added during the progress of the blow. These larger converters take a 70-ton charge, each charge requiring fifteen tons of fluxing ore, with a silica assay of 60 per cent and an alumina assay of 10 per cent. Air having a pressure of 16 lb. per square inch is used and the converters are operated by electricity. Repair to the lining of the converters is done without removing the shell from its stall. Another 60-ton crane has been installed.

THE converter slag furnace is 153 ft. long by 21 ft. wide and is placed in the converter building in the space formerly occupied by the lining department when acid-lined converters were in use. This furnace is served by one of the three cranes on the converter floor. The slag is granulated in water and conveyed by flume to the dump. The furnace will treat 650 tons of hot converter slag per day, together with 450 tons of necessary fluxes, burning 85 tons of coal. Its gases pass into a flue leading to a steel, brick-lined stack 200 ft. high. In this flue are two Stirling boilers with a rated capacity of 850 hp. each. The flue between the furnace and boilers is 15 ft. wide by 13 ft. high, and is divided into two parts, one part to each boiler, so that both or either of the boilers may be used. From the boilers to the stack the flue is 8½x6 ft. in cross-section.

CASTING or refining furnaces, small reverberatories, are on one side of the converter building within easy reach of the cranes in the converter aisle. Originally there were two: one, 14 ft. by 22 ft. 8 in., and the other, 14 ft. by 28 ft. hearth dimensions and with fireboxes 5½ ft. wide by 7 ft. long. They were fired with lump coal and provided with endless-chain anode-casting machines. Later, another furnace was built of larger dimensions, 34x14 ft., and the other two furnaces were enlarged to the same size. Pulverized coal was substituted for lump coal, giving a 50 per cent saving in fuel costs. There are two twenty-ton cranes in the casting furnace side of the building.

The main flue was built in 1903 and is connected at its upper end with a stack. The roaster plants, the blast-furnace plant, and the converter plant each had a dust chamber. Flues from roaster No. 1 and the blast furnace dust chambers were built connecting with the main flue. That from roaster No. 1 is 497 ft. long, 20 ft. wide, and 17 ft. high; whereas the blast-furnace flue is much longer, being 1,666 ft. long, 20 ft. wide, and 16 ft. high. The reverberatory has no dust chamber; its flue connects with the roaster flue and is 1,236 ft. long, 20 ft. wide, and 15½ ft. high. The old converter flue, which connects with the reverberatory flue, is 718 ft. long, 24 ft. wide, and 18 ft. high. A second converter flue was built in 1918, connecting the converter dust chamber with roaster No. 1 flue, this being 570 ft. long, 15 to 10 ft. wide, and 14 to 22 ft. deep. None of these flues have hopper bottoms. In 1916, another flue was built connecting roaster No. 2 dust chamber with the main flue; this is 387 ft. long, 25 ft. wide, and 29 ft. high, and is hopper-bottomed for the removal of flue dust. An experimental Cottrell treater was placed in this flue.

When the furnaces in roaster No. 2 were divided between zinc and copper roasting, the dust chamber was also divided, and the north 44 ft. of it, along with the old flue past the Cottrell treater, is used for zinc gases. A bypass flue was also built connecting the dust chamber with the old flue past the Cottrell treater, this bypass being 275 ft. long and 18x18.3 ft. in cross-section, hopper-bottomed.

The main flue is 2,197 ft. long and 30 ft. high. It is 60 ft. wide for 1,275 ft., and 120 ft. wide for 810 ft., with the cone joining the two, 112 ft. long. It has hopper gates spaced 10½ ft. apart, through which the flue dust is drawn into cars which are trammed by a rope-haulage system. On account of the steep hill on which the flue is built, there is a difference in elevation of 368 ft. between its lower end and the outlet into the stack, the bottom of the stack being 436 ft. higher than the tapping floor of the reverberatories. The flues are built of brick, with roofs of brick and iron. Spaced channel irons have sheet iron between them, forming an arch upon which the bricks are laid. The 120-ft. width of the main flue has a roof of boiler plates welded together. This kind of roof has proved very satisfactory if kept well painted, and to prevent excessive air leakage the brick structure of the flues is gunited every few years.

The first stack, used from 1903 to 1919, was 300 ft. high and 30 ft. inside diameter at the top. It was built of brick and tapered ¼ to ½ in. per foot. The second, or new stack, is 585 ft. high, 76 ft. inside diameter at the bottom, and 60 ft. inside diameter at the top. A little over six months was required by the Alphons Custodis Chimney Construction Company for building it, in 1918. The weight of the stack and foundation is nearly 40,000 tons. It has two openings, where the flues enter, each opening being 61 ft. wide and 60 ft. high.

Cottrell treaters of the plate and wire type were built between the main flue and the new stack, four flues connecting the main flue with the treaters, each flue serving a unit of five Cottrells. These flues are of brick and of varying lengths, 25 ft. wide by 22½ ft. high.

For the information that follows concerning copper smelting at Great Falls, Mont.; at Tooele, Utah; and at Inspiration, Ariz., I am indebted respectively to A. E. Wiggin, Manager of Reduction Works in Montana, to B. L. Sackett, General Superintendent at Tooele, and to P. D. I. Honeyman, Assistant Superintendent, International Smelting Company, at Inspiration, Ariz.

AT Great Falls, where smelting operations were all discontinued in 1917, the smelting plant originally built by the Boston & Montana Consolidated Copper & Silver Mining Company was taken over by the Amalgamated Copper Company and later merged in the Anaconda Copper Mining Company. Construction was originally begun in the early part of 1891. The plant at first had 24 Bruckner furnaces, eight tilting reverberatory furnaces, and two five-ton upright converters. The reverberatory furnaces had regenerative chambers and were fired with producer gas. A blast furnace was built in 1893. Later the plant was enlarged and the furnaces were rebuilt to meet the constant demand for larger capacity. By 1903, the Bruckner furnaces were abandoned, and eighteen six-hearth McDougal-Evans-Klepetko roasting furnaces had been added. These furnaces roasted fine copper concentrate.

In 1903, the reverberatory furnace plant consisted of five gas-fired sta-

**The Anaconda plant from the air**

tionary reverberatory matting furnaces equipped with regenerative chambers. In 1913 and 1915, two larger reverberatory furnaces were built to replace the five gas-fired furnaces. One of these furnaces was fired first with coal and later with coal dust; the other was built as a coal-dust fired furnace. From 1903 to 1918, five blast furnaces were also in operation. Converter equipment and an anode-casting furnace were also provided until the plant was closed down.

**Pouring copper anodes at Anaconda**

At Tooele, Utah, which is essentially a lead smelter, developments at the copper plant have been comparatively few, owing to the small tonnage of copper ore available for treatment.

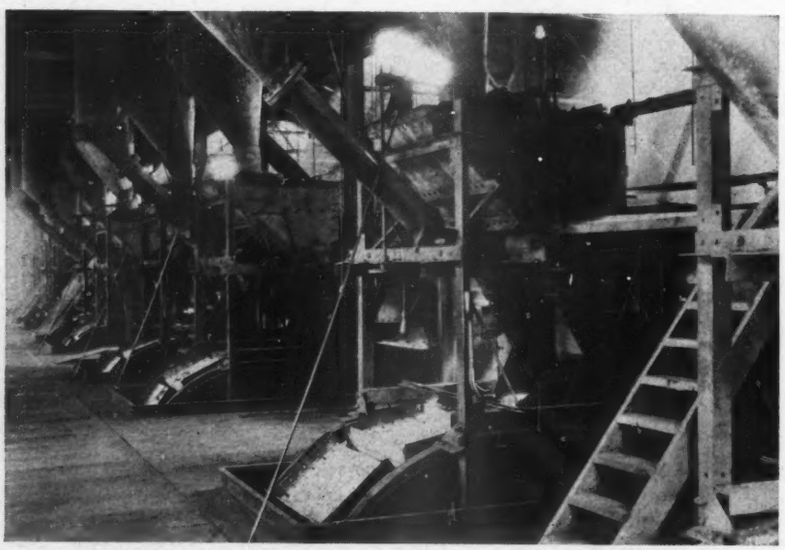
The roaster plant consists of 32 six-hearth McDougal furnaces, 16 ft. in diameter. The furnaces are coal-fired, when necessary, by grate fireboxes. In 1919 a Tooele type Cottrell treater was installed between the dust chamber and the stack, which has worked very satisfactorily both as to recovery and costs. During 1922, the character of the material treated changed from a charge made up largely of low-grade ore to one consisting largely of concentrates. The bottoms of the calcine hoppers and the tops of the calcine cars were then changed to make as nearly as possible a dust-tight connection when dumping from the roasters.

Several changes have been made in the reverberatory furnaces. In September, 1926, pulverized-coal firing was installed to replace grate firing of lump coal. This change has been the greatest forward step taken in our reverberatory practice. The furnace in operation now, 102 ft. long by 20 ft. wide, is equipped with water-cooled side jackets, seven on each side from the burner end forward, and two across the burner end. During the last six months of 1928 the charge consisted of about 30 per cent concentrate and 70 per cent of crude ore, custom matte, and copper plant secondaries. The average tonnage treated is 530 dry tons per day, with a fuel ratio of 6.2 to 1.

The coal is pulverized in a 7-ft. by 48-in. Hardinge ball mill, which has been very satisfactory and quite economical for the small amount of coal that is pulverized. At present, the tonnage pulverized is 110 tons per day, which not only suffices for the reverberatory needs but also supplies one boiler which replaces two hand-fired boilers previously used. The pulverized coal has a fineness of 70 to 75 per cent minus 200 mesh and 8 to 10 per cent plus 100 mesh.

The converters are of the horizontal, cylindrical type, with 8x12½-ft. shells having a capacity for about a ten-ton charge of matte. In 1917, a pipe type Cottrell treater was put into operation on the gases from the copper converters. These gases carried quite an amount of fume high in lead, so, on account of the necessity for humidification of the gases treated, and on account of their high acid content, the cost of operating the treater was high and the recoveries were low. During the summer of 1928, an operation worked out for lead matte converting was extended to the copper converter operations, whereby acid neutralization was taken care of by the Sprague process (addition of lime to the gases) and the subsequent treatment of these gases in a baghouse instead of in a Cottrell treater. To accomplish this, it was necessary to change the arrangement of the flue system and build an additional flue so that the gases could be cooled before entering the baghouse. This has eliminated the high cost of Cottrell operations at this plant, caused by the factors cited, and has given a satisfactory recovery.

**Sintering equipment for lead ores at Tooele**



The Miami plant of the International Smelting Company, in Arizona, is treating the entire output of concentrates of the Globe-Miami district, the big producers being Inspiration Consolidated Copper Company, Miami Copper Company, and Old Dominion Company. As well, the smelter handles a considerable tonnage of siliceous ores, mined locally, and used as converter flux.

Concentrates are received in steel cars, which are weighed on standard track scales and run onto the unloading pocket, where they are sampled and unloaded. In sampling concentrates, a 1½-in. auger is used and 24 holes are put down in each car for the full depth of the concentrates. Moisture samples are taken on three vertical faces while unloading.

Concentrates are unloaded directly onto a conveyor belt and transported to the bedding bins, where the furnace charge is made up. With the exception of converter secondaries, no flux is added to the beds. The bedding plant consists of three 4,000-ton bins, each of which is fed by a belt conveyor and tripper car traveling up and down the length of the bed, automatically spreading the charge. Beds are reclaimed through the bottom of the bins, onto a conveyor which feeds directly to the charge cars. Charge cars consist of former concentrate cars, a sufficient number of which have been put into this service to insure a flexible storage of

furnace feed. At the present rate of operation, one shift reclaims sufficient charge to keep two furnaces running for 24 hours.

The reverberatory plant consists of four furnaces, all 120 ft. long. Three of them are 21 ft. wide, but the fourth has a width of 25 ft. Wet charge, containing 10 to 13 per cent moisture, is fed directly to the furnaces. Various methods of charging have been worked out, and at present, of the two furnaces operating, one is fed by means of a drag chain conveyor, whereas the other has a belt-conveyor charging system. The drag-chain conveyor is fed from a hopper by an apron feeder and the wet feed is dragged along a conveyor box, equipped with suitable liners, and discharged to the furnace through a series of charge pipes spaced down the length of the charging zone. The belt conveyor system utilizes a "shuttle conveyor" which moves up and down over the charging zone, feeding the furnace through the various charge pipes. This entire belt system is electrically and automatically controlled in such a way that only a predetermined amount of charge is fed in through each hole, according to the smelting rate of the furnace at that particular hole.

The daily smelting rate is from 500 to 600 tons of wet feed per furnace. Fuel ratios are necessarily high and run from 0.80 to 1 barrel of oil per ton of dry charge. However, waste-heat recoveries are also high, being in the neighborhood of 50 per cent.

The furnaces are oil-fired and burn from 420 to 450 bbl. of oil per furnace day. There are ten burners of the Cananea type, per furnace, using 40-oz. air for atomization. Combustion is carefully controlled, gas analyses being taken daily. Furnaces operate on about 0.05 in. of draft, taken through the verb arch. Converter slag is charged molten to the furnace through a launder which enters through the middle of the firing-end wall, just under the arch.

Both furnaces are skimmed and tapped at the front or uptake end. Matte tap holes are just back of the verb arch in the side walls. Slag is tapped through a small hole in the adobe breast, and from six to eight pots are skimmed per shift. The matte is tapped as needed by the converter plant into a twenty-ton ladle which rides on a ladle car, which is then trammed around to the converter plant.

The furnace charge averages 30 per cent copper and produces a 40 to a 44 per cent matte. The final dump slag is somewhat basic and has the following analysis: copper, 0.50 per cent; silica, 31 to 33 per cent; alumina, 8; ferrous oxide, 50 to 54; and calcium oxide, 1.5 per cent.

The waste-heat boiler plant consists of ten Stirling type boilers. Seven of these are 713 hp., whereas the remaining three are of 850 hp. Normally, three waste-heat boilers handle the gases from each furnace. Exit gases are around 550 deg. F.

In the converter plant are five 12-ft. Great Falls converters and one 12x 20-ft. Peirce-Smith converter, together with two 60-ton tilting copper furnaces and casting machines. Converters are served by two 40-ton Morgan cranes. Converter practice is standard and about 70 to 80 tons of copper are produced per converter day. At present, the plant is producing about 200 tons of copper per day.

One interesting feature worthy of mention is the production of anodes from blister copper. These anodes are used to make starting sheets at the Inspiration leaching plant, around 600 tons per month being cast. The casting arrangement is a line of stationary molds, down the length of which travels a tilting ladle car. Anodes weigh about 1,050 lb. and are handled by a small Shepard monorail crane.

Ladle skulls and converter cleanings are handled in the "skull breaker," where they are broken up and passed through a grizzly into a car below. Converter gases pass from the converters through gooseneck flues to the Cottrell plant, where the dust is precipitated in pipe treaters.

# Promoting Sales of the Product

By J. N. McDonald

IN THE YEAR 1921 a general depression in the copper industry followed the abnormal war-time demand for copper. Large surplus stocks of copper were on hand, as well as large tonnages of sheets, tubes, and other commercial shapes. In addition, the scarcity of copper during the war period resulted in the acceptance of substitutes, and these markets temporarily lost to copper were not immediately recoverable. Recognition of these conditions led to the formation, by the leading copper producers and fabricators, of the Copper & Brass Research Association, which has done much in reviving old and developing new uses for copper, brass, and bronze.

Upon the acquisition of The American Brass Company, with its direct contacts with the markets for copper and copper alloys, it was practicable for Anaconda to undertake direct sales promotional work not previously possible, and a campaign to increase the use of Anaconda copper and copper alloys was commenced. The first step was the selection of a trademark to identify all Anaconda products. The now well-known Spear Head trademark —Anaconda—From Mine to Consumer —symbolical of copper's long service to humanity and Anaconda's service to industry and the consumer, was adopted. This mark was applied to all products of The American Brass Company and of other Anaconda units.

After a thorough survey of conditions in the various markets, it was decided to concentrate the principal effort on the building industry. Two factors influenced this decision; namely, that substitutes had replaced copper to a great extent in this market, and many Anaconda products used in building could be readily identified by the ultimate consumer. The methods employed to promote the use of copper and brass in this field are best illustrated by the campaign used in developing the market for brass pipe.

Intensive selling backed by advertising in all its phases was used to promote the sale of Anaconda brass pipe. Advertising was placed in architectural journals and plumbing publications, and followed by the distribution of informative booklets and mailing pieces presenting the advantages of Anaconda brass pipe to these important groups. Acceptance of brass pipe by builders and house owners was accomplished by advertising in national magazines and newspapers and by mailing booklets and folders. Plumbing contractors were furnished window displays, signs, folders, blotters, and other devices, to help them sell Anaconda brass pipe. The American Brass Company was represented at all important trade conventions, and a motion picture illustrating the manufacturing of brass pipe was shown before many groups. Exhibits were also used as a further means of contacting the various trade factors.

That this program has been effective is best attested by a comparison of the sales figures for Anaconda brass pipe in 1920 and 1928. During that period an increasing volume of sales was built up, until in the latter year seventeen times as much brass pipe was sold as in 1920. Today brass pipe is used extensively in all sections of the country, and the volume is growing in spite of a falling off in construction.

Promotion work, similar to that used for brass pipe, has been undertaken to stimulate the use of Anaconda sheet copper for roofing, gutters, and leaders, and bronze wire for screen cloth; and considerable advertising has been devoted to the economy of rust-proof construction generally. Special promotion work has also been devoted to extruded bronze shapes, to Tobin bronze for welding, to the free-turning brass rod, and to special alloys such as Everdur.

A large part of the success achieved by sales promotion activities is attributable to the effective manner in which the Anaconda technical departments have solved problems arising from the use of non-ferrous metals and to the service which the Anaconda organization renders industry in furnishing copper and copper alloys of uniform high quality, which is safeguarded by the co-ordinated control of all manufacturing operations, from mine to consumer.

# Refining Anaconda Copper at Raritan and Great Falls

By W. T. BURNS
*Manager, Raritan Copper Works*

TWO copper refineries are maintained by the Anaconda Copper Mining Company, one on the Eastern seaboard, at Perth Amboy, N. J., and the other about 200 miles from the company's mine and smelter operations in Montana, at Great Falls. Though the Great Falls plant is somewhat older, the Eastern plant is larger, and will be described first.

The Raritan Copper Works, as the Perth Amboy plant is known, is situated at the mouth of the Raritan River, 23 miles from New York City. It is the outgrowth of the old New England Electrolytic Copper Company, of Central Falls, R. I., owned and operated by the Lewisohn brothers until August, 1900, when it was dismantled. Construction of the first unit of the Raritan works, with a capacity of 120,000,000 lb. per year, was completed by the Lewisohn brothers, and production started in March, 1899. It was operated by the builders until February, 1900, when it was taken over by the Anaconda interests, which have since owned and operated the plant. Since operations began, the works have been enlarged and rebuilt, with the result that the present capacity is four and a half times what it was originally, or 45,000,000 lb. of cathodes per month, and a 57-acre tract on the waterfront is occupied.

Crude copper is received from Africa, South America, Mexico, and the United States in the form of pigs or slabs varying considerably in composition. All but 6 per cent of this copper comes by water and is delivered at the dock on company barges, from which it is transferred to industrial cars and, after weighing and sampling, delivered to the furnaces.

Four reverberatory anode furnaces are provided, data concerning which are given in Table I. The furnaces are charged by two cranes, each of 5,000-lb. capacity. Some oxidizing and poling is necessary, but in general little refining is done in these anode furnaces. One charge is taken out each 24 hours, the

**Table I—Anode Furnace Data**

| Furnace Number | Size of Hearth, Feet | Output per Charge, Pounds | Waste-heat Boilers—Type | Waste-heat Boilers—Horsepower Rating |
|---|---|---|---|---|
| 4 | 43 x14 | 650,000 | Stirling | 600 |
| 5 | 38½x14 | 500,000 | Stirling | 400 |
| 7 | 50 x14 | 750,000 | Babcock & Wilcox | 450 |
| 8 | 45 x14 | 750,000 | Stirling | 400 |
| | | 2,650,000 | | 1,850 |

molten copper being cast in straight-line-type casting machines, equipped with cast-iron anode molds. The anodes for the commercial refining tanks are 37 in. long, 28 in. wide, and 1⅛ in. thick, and average 525 lb. each, but the anodes for the stripper section are slightly larger. The average rate of casting is 80 tons per hour. Table II gives a typical analysis of the anodes produced.

**Table II—Analysis of Anodes**

| | |
|---|---|
| Copper, per cent | 99.25 |
| Silver, ounces per ton | 30.0 |
| Gold | 0.40 |
| Arsenic, per cent | 0.060 |
| Antimony | 0.022 |
| Nickel | 0.050 |
| Lead | 0.053 |
| Iron | 0.058 |
| Bismuth | 0.003 |
| Sulphur | 0.004 |
| Selenium | 0.048 |
| Tellurium | 0.038 |
| Oxygen | 0.100 |

Originally, the fuel for the anode furnaces comprised lump coal burned on grates in a firebox; later, fuel oil replaced the coal; today, powdered coal is used with entire satisfaction on these as well as on all other copper furnaces in the plant. The average coal consumption over a period of six months was 210 lb. per ton of anodes, using West Virginia coal of the proximate analysis shown in Table III.

The electrolytic refinery, to which the impure copper from the anode furnaces is transferred, consists of two sections known as Nos. 1 and 2 tank houses. No. 1 contains 1,800 cells or tanks, and No. 2, 1,656, a total of 3,456, of which 3,180 are for commercial copper and 276 for starting sheets. These are arranged on the Walker system in nests of eleven and thirteen cells each. The general arrangement is known as the multiple system, the electrodes in each tank being in multiple and the tanks in series.

The tanks are 9 ft. 11 in. long, 2 ft. 10 in. wide, and 4 ft. deep. Most of them are built of wood and lined with ⅛ in. of 6 per cent antimonial lead sheeting, though during the last year a reinforced lead-lined concrete tank has been developed that will hereafter be

**Table III—Proximate Analysis of Coal Used**

| | |
|---|---|
| Moisture, per cent | 0.70 |
| Fixed carbon, per cent | 59.50 |
| Volatile matter, per cent | 31.50 |
| Ash, per cent | 8.00 |
| Sulphur, per cent | 1.10 |
| B.t.u. dry | 14,000 |

used in replacements. Anodes are placed in the tanks, and cathodes removed therefrom, by eight electrically operated cranes in each tank house.

The starting sheets are deposited on plates of rolled copper suspended between anodes in the stripper tanks. There are 25 anodes and 24 blanks in each tank. After a current of 7,000 amp., equivalent to 17 amp. per square foot, has deposited copper on these blanks for 24 hours, the crane removes one-half of the blanks from a tank, the remaining blanks carrying the full current, to racks where the sheets are stripped from the blanks. The blanks are then greased to prevent the deposited copper from adhering to the blanks, supporting loops are riveted to the sheets by a special machine, and the sheets are ready for the tanks. At full production, 300,000 of these sheets are required per month.

Twenty-eight anodes and 29 starting sheets are placed in each commercial tank, the anodes being suspended by their lugs, and the starting sheets by copper rods inserted into the loops. The electrolyte is heated in hot wells by lead pipes through which steam, extracted from the turbines at 2-lb. pressure, is passed. It is circulated, at 4 g.p.m., through a cascade of two cells by Antisell centrifugal pumps, constant circulation being maintained to prevent stratification. Table IV gives its composition.

As the purity of the deposited copper is dependent largely on the purity of the electrolyte, it is necessary to keep

**Table IV—Composition of Electrolyte**

(In grams per liter)

| | |
|---|---|
| Copper | 45. |
| Arsenic | 12.5 |
| Antimony | 0.4 |
| Iron | 1.20 |
| Nickel | 8.5 |
| Free sulphuric acid | 200. |
| Chlorine | 0.03 |
| Specific gravity | 1.260 |

the electrolyte-purifying department in continual operation. The purifying is effected by a series of crystallizations in which copper sulphate and nickel sulphate (single salt) are produced and marketed. Constant attention to details is essential in the electrolytic refining of copper to insure a product of the highest grade.

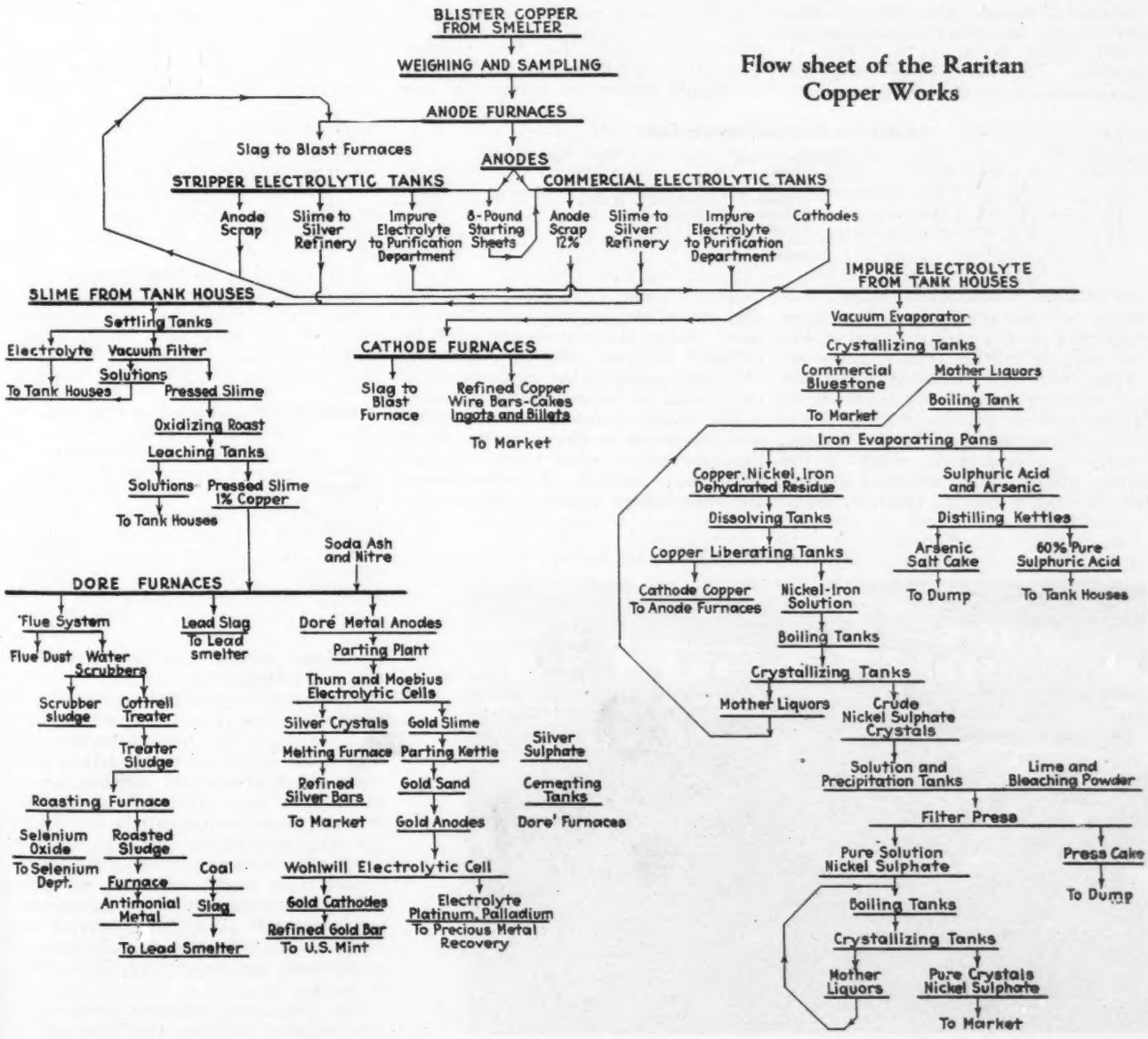
With a current of 8,000 amp., equivalent to 18 amp. per square foot, the starting sheets will have attained a weight of 158 lb. each in ten days, whereupon they are drawn as finished cathodes, washed, and sent to the furnaces for melting, new starting sheets replacing them in the cells. This procedure is repeated twice, whereupon the skeletons of the original anodes are also drawn. In this manner three 158-lb. cathodes and one piece of 75-lb. anode scrap are produced each thirty days from one 525-lb. anode and three 8-lb. starting sheets. Under these conditions an average ampere efficiency of 93 per cent is obtained, with a production of 8.6 lb. of copper per kilowatt-hour.

The insoluble portion of the anode settles to the bottom of the cell, forming a silver- and gold-bearing slime that is emptied out by removing a lead plug in the bottom of the tank. It is then conducted, by a system of launders in the basement of the building, to a collecting tank, from which it is pumped to the silver refinery for the recovery of the precious metals.

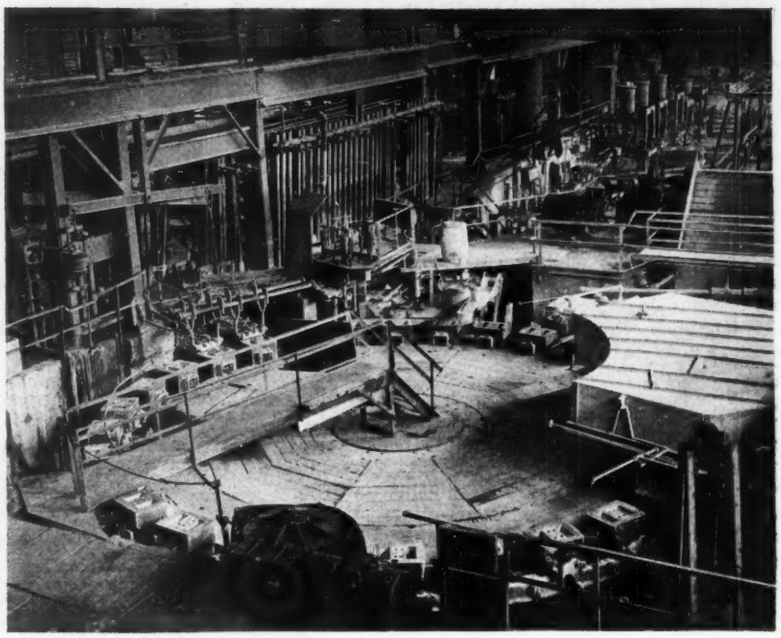
In connection with the description of the electrolytic department, a few words should be said about the generation and distribution of power. The power plant has been reconstructed and now consists of three 1,200-hp. boilers, operating at 365-lb. pressure, and delivering steam at 660 deg. F. These boilers are normally operated at 180 to 200 per cent of rated capacity. They supply steam to three turbo-generators, two of 5,000 kva., and one of 3,125 kva., all delivering power at 2,300 volts and 60 cycles. The turbines are arranged for condensing operation with bleeder points at 2-, 30-, and 165-lb. pressure to supply steam for the diversified plant operations. From the old equipment, four 763-hp. boilers have been retained, and four engine-driven direct-current generators for reserve power. The alternating current is taken to the tank houses, where it is converted into direct current of the proper voltage for refining.

No. 1 tank house is divided into two electrical circuits of 900 cells each. In No. 2 tank house are two electrical circuits of commercial cells, one of 660 and one of 720 units; also one circuit of 276 stripper cells in which the starting sheets for both tank houses are made.

In No. 1 tank house substation are two 2,400-kw. motor-generator sets, each direct-connected to two 300-volt



4,000-ampere direct-current generators. Each of these units supplies current to one electrolytic circuit of 900 cells. In No. 2 tank house substation are two 1,920-kw. sets, each connected to two 240-volt, 4,000-ampere direct-current generators, one of these units furnishing current to a circuit of 660 refining cells and one to a circuit of 720 cells. Also at this tank house is a 1,076-kw. synchronous rotary converter, equipped for automatic constant current control, normally 7,000 amperes, 100 volts. This unit provides direct current for the stripper section of 276 cells.



Casting side of the Raritan copper refining furnaces

CATHODE furnaces to the number of four, of the reverberatory type, are provided. In these the cathodes are melted and cast into wirebars, cakes, and other commercial shapes.

These furnaces are charged by two electrically operated charging machines, one having a capacity of 5,000 lb. and the other of 7,500 lb. Charging occupies two hours, melting ten, oxidizing five and a half, poling two, and casting four and a half hours.

Though sometimes referred to as "refining furnaces," this term is a misnomer, as the cathodes charged are really higher in copper than the cast product. The function of these furnaces is merely to convert the cathodes into castings of commercial shape. But owing to the presence of entrapped electrolyte on the cathodes, and to the fact that, in melting copper, gases resulting from the combustion of the fuel are absorbed, what appears to be a simple melting process in reality becomes more complex. In the furnace treatment advantage is taken of the strong affinity of the absorbed impurities for oxygen and the relatively lower affinity of copper for that element. From 0.03 to 0.05 per cent of oxygen is always present in commercial cast copper. Table VI gives a typical analysis of the product of these furnaces, and is also representative of the cathodes charged, with the exception, as already noted, of the oxygen content, there being no oxygen in the cathodes.

**Table V—Cathode Furnace Data**

| Furnace Number | Size of Hearth | Output per Charge, Pounds | Waste-Heat Boilers — Type | Waste-Heat Boilers — Hp. Rating |
|---|---|---|---|---|
| 10 | 44 ft. 2 in. x 17 ft. 2 in. | 620,000 | Babcock & Wilcox | 450 |
| 11 | 33 ft. 8 in. x 17 ft. | 510,000 | Babcock & Wilcox | 450 |
| 12 | 37 ft. 6 in. x 17 ft. | 575,000 | Stirling | 600 |
| 14 | 37 ft. 6 in. x 17 ft. | 575,000 | Stirling | 600 |
| | | 2,280,000 | | 2,100 |

The reverberatory furnaces are lined with magnesite brick up to the metal line; above this, silica brick are used, as well as in the roof. Furnace bottoms are either silica sand or silica brick.

Interior of No. 2 tank house at Raritan

The Clark casting machine used consists of a wheel 36 ft. in diameter, with molds arranged radially. These molds are of copper and are sprayed with a bone-ash emulsion to produce a smooth surface on the castings. The castings are removed from the bosh, into which they are dumped, by a chain conveyor, on which they are inspected; they are then transferred, by air hoists, to railway cars for shipment.

The fuel ratio of these furnaces, including the coal used in heating cold furnaces and in carrying over furnaces for a day or more when there is insufficient copper available for charging them, averages 250 lb. of coal per ton of copper, or slightly higher than for the anode furnaces.

**Table VI—Composition of Cast Electrolytic Copper**

| | Per Cent |
|---|---|
| Copper | 99.94 to 99.97 |
| Oxygen | 0.02 to 0.05 |
| Silver | 0.0010 |
| Gold | 0.00001 |
| Sulphur | 0.0020 |
| Iron | 0.0025 |
| Nickel | 0.0015 |
| Arsenic | 0.0015 |
| Antimony | 0.0015 |
| Average conductivity annealed wire | 100.70 |

Three smaller powdered-coal-fired reverberatory furnaces in a separate building are used for the production of copper and phosphor copper billets and other copper alloys, and cadmium wirebars. They have a capacity of 125,000 lb. each, with hearths 10x20 ft. They are entirely hand-operated and are not provided with waste-heat boilers. Charging of these is by means of the old-fashioned paddle and the charge is ladled out with 20-in. ladles carried on trolleys. The billets run from 2½ to 8 in. in diameter and weigh from 70 to 600 lb. each.

The slag and cobbings from the reverberatory furnaces (the slag being

equivalent to about 2½ per cent of the output of the furnace), and from the doré furnaces in the silver refinery, are treated in a 42x90-in. blast furnace having a daily capacity of 150 tons of charge. From this the metal is tapped directly into water-cooled cast-iron molds, producing 3,500-lb. pigs. The slag overflows from a settler into a cast-iron launder, where it is granulated by a stream of salt water and then pumped to the dump.

The silver refinery for treating the anode slime consists of the slime plant, the doré plant, the parting plant, and the gold plant. At the slime plant, the slime for treatment in the doré furnaces is prepared. The mixture of slime and electrolyte received from the electrolytic department is settled in four large settling tanks, the clarified electrolyte being returned. The slime is then filtered, using an Oliver filter, after which it is given a light roast in a small oil-fired roasting furnace to break down the selenides and tellurides and to oxidize the copper. Then the roasted slime passes to agitators operated with mechanical paddles, where it is leached with hot dilute sulphuric acid to remove the copper, and, after washing and filtering through a Moore filter, is sent to the doré furnaces. These are small oil-fired reverberatories in which the slime is melted and by oxidation and the use of proper fluxes the rest of the impurities are removed. The resulting metal, known as doré, contains 98.5 per cent silver and 1.5 per cent gold, and is cast into anodes for the electrolytic separation of the silver in the parting plant.

Two types of electrolytic cells are used in the parting plant, the Moebius cell and the Thum cell. In the Moebius system the anodes are inclosed in canvas bags to catch the gold slime and are suspended vertically in a deep mastic-lined concrete tank. The silver is deposited on cathodes of stainless steel, from which the deposited crystals are removed by a mechanical scraper, falling into trays, from which they are periodically removed. In the Thum system a shallow tank is used, the bottom of which is covered with graphite slabs forming the cathode. The anodes are placed horizontally in a wooden frame covered with canvas. Silver is deposited on the cathode in a loose crystalline form and is periodically removed. The electrolyte in both systems consist of a neutral solution of silver and copper nitrate containing 60 grams per liter of silver and 40 of copper. All deposited silver is removed, washed, melted in an oil-fired retort, and cast into bars having a fineness of over 999. The gold slime remaining in the canvas covering is collected and sent to the gold plant, where it is boiled with sulphuric acid to remove the silver, washed, melted, and cast into anodes for the Wohlwill process. This process consists in electrolysis, using a hot acid solution of gold chloride, the gold being deposited on thin rolled-gold cathodes, which are periodically removed, washed, melted, and cast into bars with a fineness of over 999. Any platinum or palladium present in the gold anodes passes into the electrolyte, and when a sufficient concentration has been reached, is recovered by ordinary chemical methods of precipitation with ammonium chloride.

The silver refinery has a capacity of 2,500,000 troy ounces of silver and 25,000 troy ounces of gold per month. Electrolytic slime from the Great Falls refinery and the doré bullion from the East Chicago plant of the Anaconda company, together with the electrolytic slime from the Raritan Copper Works, are treated in this refinery. In addition to the silver and gold produced, small amounts of selenium, tellurium, platinum, and palladium are recovered.

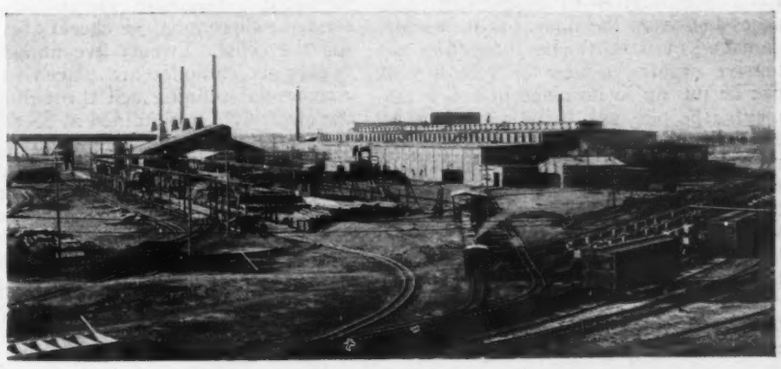
PULVERIZED coal is used for power generation at the Raritan works and for firing all of the copper furnaces. The coal is delivered by rail on a trestle and dumped into pockets, from which it is fed through a crusher and transported by conveyors to elevated storage bins in the coal-pulverizing plant. Here there are four pulverizing mills each served by its drier, fans, and collectors; and a system for transporting the pulverized fuel to the various points of consumption. From the storage bins the crushed coal is gravity-fed through waste-heat driers utilizing the heat in the power boiler flue gases. These are drawn through the driers by fans, returned to the main flue, and thence go to the stack. Leaving the driers, the crushed and dried coal is fed to impact-type pulverizers, of the air-swept type, and of seven tons' per hour capacity each when pulverizing to 70 per cent plus 200 mesh. A screw conveyor transfers the pulverized fuel from the mill and drier collector to two "shooting" tanks in which the coal is weighed and from which it is transported by compressed air after first having been thoroughly aërated. Electro-pneumatic valves govern the transport of the coal from either "shooting" tank to any point of consumption. At the boiler the fuel is taken from storage bins and fired through horizontal burners by screw feeders and flare-type blowers. Fuel for the copper furnaces is delivered to substations at a safe distance from the furnaces, and there, under the control of the screw feeders, is mixed with air and fed to the furnaces through horizontal burners.

## *The Great Falls Copper Refinery*

The first Great Falls refinery was built in 1892 by the Boston & Montana Consolidated Copper & Silver Mining Company, to refine the product of the Great Falls Reduction Works of that company. The mines were in Butte, 180 miles distant, the ore being shipped to Great Falls for concentration and smelting. This refinery was among the several early electrolytic refineries of this country, but it was unique in several respects: it was the only refinery in this country that treated anodes that were the direct product of the converters; and it was also the only refinery that operated at a current density of 40 amp. per square foot or anything approximating this density. This high density was the result of cheap power and a desire to obtain a large output from a small plant. Direct-current generators were used, direct-connected to water wheels, situated 2,000 ft. from the tank house, the current being conducted over a transmission line containing 668,000 lb. of copper.

The tank house contained 288 lead-lined wooden tanks, 9 ft. 7 in. long, 2 ft. 4 in. wide, and 3 ft. 9 in. deep, inside measurements. The tanks were arranged in cascades of eight units, with aisles between the double rows. Twenty anodes and twenty cathodes, connected in multiple, comprised each cell, and all the cells were in series and in one circuit. With 10,000 amp., equivalent to 40 amp. per square foot of cathodes, two-day cathodes, averaging 54 lb. each, were produced at 85 per cent ampere efficiency at the rate of 3.5 lb. of copper per kilowatt-hour. The maximum capacity of the plant was 4,000,000 lb. per month. Operating under these conditions, with rough blister anodes low in copper and high in impurities, it is remarkable that high-grade wirebar, reported in those days in terms of conductivity on hard-drawn wire only, averaging 98.5 per cent conductivity, was obtained. It must be admitted, however, that good wirebar was obtained only as the result of eternal vigilance

**The refinery at Great Falls, Mont.**



and some sleepless nights. The art of electrolytic refining was new and the nearest laboratory where physical and electrical tests were made was in Ansonia, Conn. Consequently, it was necessary to wait from two to three weeks for the conductivity results that can be had today before the charge is one-half out. It is interesting to note that the cost to produce wirebar from converter copper in this crude, hand-operated, high density plant was far below the cost obtained in the modern plant of today.

The cathodes were melted in small reverberatory furnaces of fifteen to twenty tons' capacity, the copper being ladled out of the furnaces with 9-in. hand ladles and carried to the molds by the workman, who was known as a copper dipper, a very important individual in those days. Now, 37 years later, the practice of melting cathodes and casting shapes remains unchanged, excepting for the fact that larger furnaces and improved mechanical appliances are in use. The metallurgical treatment of the copper remains the same.

This plant continued in operation until the present refinery at Great Falls was built. The last copper was drawn May 9, 1917, after which the old refinery was dismantled after an honorable career of 25 years.

During 1910, the properties of the Boston & Montana Consolidated Copper & Silver Mining Company were taken over by the Anaconda Copper Mining Company. In 1915, the present refinery was built, and it produced the first copper in March, 1916. The plant has operated continuously since and has treated only the product of the Anaconda reduction works.

As originally constructed, the tank house was 383 ft. long and 252 ft. wide and contained 1,020 refining cells. In 1926 it was increased 50 per cent in size; the building was increased to 535 ft. in length and the number of cells increased to 1,530, of which 90 are stripper cells.

This plant is built on the so-called two-cell system; that is, a group of single cells is arranged in double rows of ten, five in a cascade, with aisles between each double row. Other refineries use the Walker system, where the tanks are built in nests, which results in an economy of floor space and a saving in power losses in conductors, compared with the two-cell system.

With the system installed at Great Falls each tank is an independent unit, accessible from the floor. If it becomes damaged or is otherwise in need of extensive repairs, a new or rebuilt tank can be put up in its place in a few moments, the old tank being picked up by the crane, put on an industrial car, and moved to the shop, where the necessary repairs are made. Worn tanks are continually being replaced with new ones, so that no wholesale replacing of old tanks need be undertaken at any one time.

Great Falls plant was designed to operate at high density in a locality where power was cheaper and labor was more expensive than in the East. The two-tank system was therefore adopted and the eleven years' operation of the plant, at current densities of from 15 to 28 amp. per square foot, have justified the course taken. The plant is now operating at 28 amp. per square foot. Under these conditions an ampere efficiency of 94 per cent can be maintained, which I do not think would be possible with any other arrangement of the tanks.

The tanks are of wood, lead-lined, 10 ft. 3 in. long, 2 ft. 10 in. wide, and 3 ft. 9 in. deep, inside dimensions. Several years ago a reinforced monolithic concrete tank, lead-lined, was placed in service, which has proved to be entirely satisfactory, so all replacements are now made with this type.

The tank house is divided into twelve sections of 120 cells each, and the stripper section of 90 cells. Each section has a separate electrolyte, which is pumped by Pohlé air lifts operated by a compressor in the substation basement. The pipe columns are of cast lead-and-antimony pipe, 6 in. inside diameter, with $\frac{3}{4}$-in. walls. The lift is 10 ft. and the submersion 20 ft. Owing to the extremely low temperatures prevalent at times, the roof of the tank house is provided with a dead air space and the windows are double glazed to prevent condensation.

The building is divided into four crane bays of 360 tanks each. There are seven, 60-ft. span, 10-ton cranes. A crane transfer at one end of the building makes it possible to transfer cranes from one bay to another. The 90 stripper tanks are not under the cranes.

Anodes are cast at the Anaconda smelter, rather than at the refinery, and are of the coped lug type, weighing 630 lb. each. They are received in box cars at Great Falls, where they are transferred to rack cars by special equipment, each rack car holding twenty-five anodes, the number required for one refining cell. They are of the approximate analysis shown in Table VII.

**Table VII—Approximate Analysis of Anodes**

| | |
|---|---|
| Copper, per cent | 99.3 |
| Arsenic, per cent | 0.085 |
| Antimony, per cent | 0.04 |
| Silver, ounces per ton | 67.0 |
| Gold, ounces per ton | 0.3 |

The starting sheets are made in a separate section of ninety cells by vertical stripping over the tanks. Thirteen-hour sheets, weighing 7 lb. each, are produced. To these sheets loops are riveted, whereupon the sheets are ready for the cells. Twenty-five anodes and twenty-six cathodes are placed in each commercial refining cell through which the electrolyte, maintained at 55 deg. C., is circulated at the rate of 5 g.p.m. Under these conditions, with a total current of 10,000 amp., the current density is 26.4 amp. per square foot and the production of the plant 27,000,000 lb. per month, with an average ampere efficiency of 94 per cent. Production per direct current kilowatt-hour is 6.0 lb. The age of the cathodes is six days, and the anodes remain in the cells for 24 days. When finished, the weight of the cathode is 140 lb. and the anode scrap averages 9.5 per cent.

The electrolytic slime, containing the gold and silver, is removed from the bottom of the cell every 24 days and pumped to the slime plant, where it is leached for the removal of the copper, dried, and packed for shipment to the Raritan Copper Works for refining. Analysis of this slime is given in Table VIII.

**Table VIII—Analysis of Leached Great Falls Electrolytic Slime**

| | |
|---|---|
| Copper, per cent | 1.5 |
| Arsenic, per cent | 5.0 |
| Selenium, per cent | 2.0 |
| Tellurium, per cent | 15.6 |
| Lead, per cent | 4.6 |
| Sulphur, per cent | 2.7 |
| Silver, ounces per ton | 13,300 |
| Gold, ounces per ton | 75 |

Purification of the electrolyte is effected by diverting daily to the purification plant a sufficient amount of the working electrolytes to prevent building up of the impurities. This solution is concentrated by boiling to 48 deg. Bé. and run to crystallizing tanks, where about 80 per cent of the copper is recovered as bluestone, which is redissolved and returned to the tank house. The mother liquor remaining, containing the acid and impurities in the concentrated solution, is treated in insoluble-anode tanks for removal of the remainder of the copper and the arsenic, antimony, and bismuth. The slime produced in these tanks is sent to the converters and the solution is further concentrated by boiling to 56 deg. Bé., when, upon cooling, the iron, nickel, lime, magnesia, and other materials are precipitated as basic salts and wasted. The acid, freed of impurities, is returned to the tank room. The importance of keeping the copper as free as possible of impurities may be illustrated by the fact that the normal arsenic content of high conductivity copper does not usually exceed 0.0015 per cent, and if this impurity, with the normal quantities of other impurities present, reaches 0.0035 per cent, the conductivity would be so seriously impaired that the copper could not be marketed.

The tank house is divided into three electrical circuits of 540, 510, and 480 tanks, respectively. Power is purchased from the Montana Power Company at 6,600 volts. The refinery substation contains seven synchronous motor-generator sets, each consisting of one 1,730-hp., 6,600-volt a.c. motor driving two 600-kw. 200-volt, 3,000-amp. d.c. generators, which furnish the power for the electrolytic circuits. In this substation are also the various units for supplying electric power and compressed air for the general operation of the plant.

The casting department handles all of the cathodes, after weighing, except those that are shipped to the consumer in the cathode form. The casting building is near the tank house and contains three reverberatory furnaces. Two of these have hearths 45 ft. 7 in. x14 ft. 4 in.; the third has a hearth 45 ft. 7 in.x13 ft. wide. Two of

the furnaces are equipped with 400-hp. Stirling waste-heat boilers, while the third has a 300-hp. boiler of the same type. Each furnace has a capacity of 560,000 lb. of copper. A charging crane, of the suspended type, with a capacity of 7,000 lb., serves the three furnaces.

As it is not possible to charge the entire 560,000 lb. of high-density cathodes at one time, 400,000 lb. is put in at the first charging. After the furnace has been under heat for a few hours, it is opened up and the remaining 160,000 lb. is charged. After melting and the customary oxidizing and poling, the furnace is ready, usually in the morning of the following day, for tapping. The copper is cast on a modified Clark casting machine, somewhat similar to the Raritan wheel, but unlike the Walker wheel in general construction and arrangement of the molds.

The rates of casting at Great Falls are shown in Table IX.

These furnaces now use natural gas for fuel and are the only important copper-refinery furnaces that have been operated successively with coal burned on grates, pulverized coal, fuel oil, and natural gas. In Table X are shown the results obtained while using these various fuels, in terms of the amount of fuel used and the heat units required, per ton of copper.

**Table IX—Casting Rates in Pounds Per Hour**

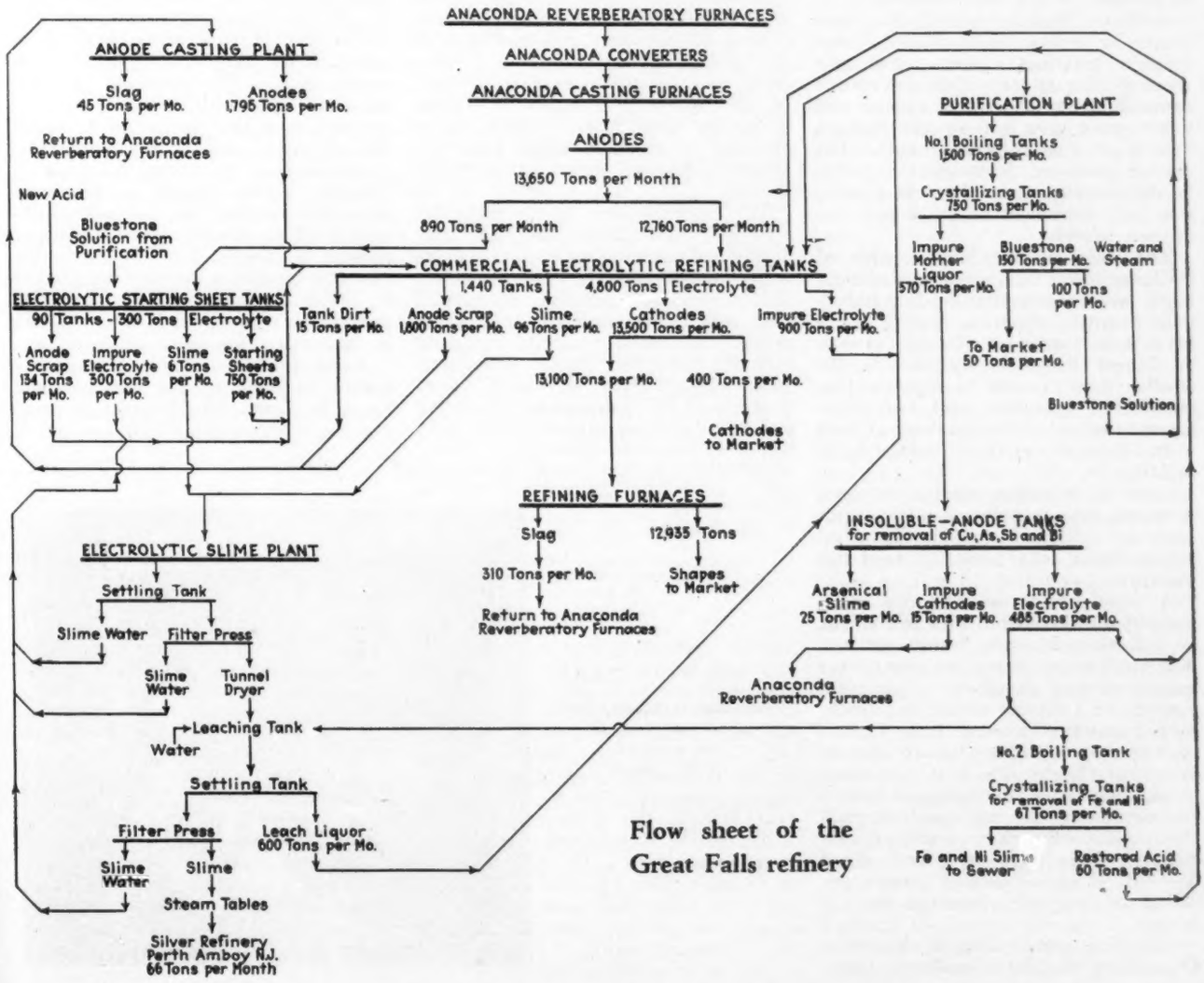
| | |
|---|---|
| Wirebars | 150,000 |
| Ingots | 100,000 |
| Double ingots | 120,000 |
| Wedge cake | 95,000 |
| Square cake | 80,000 |

Choice of fuels depends on their cost. Not many years ago Great Falls was using powdered coal and Raritan was burning oil. A few years later Great Falls found it advantageous to change to oil and about the same time Raritan, for economic reasons, changed from oil to powdered coal. All of the various fuels used at both of these plants have been satisfactory so far as the quality of the product is concerned. The powdered coal results in an increase in the amount of slag produced, but this does not retard the action of the furnace; it is only a condition to be taken into the cost consideration. At Raritan it was found that the furnaces actually worked faster with powdered coal than with oil, making it possible to take out seven charges per week, if desirable, which was not possible when fuel oil was used. Because of the high affinity of sulphur for copper, the coal and coke used on the furnaces should be low in this impurity, not exceeding 1 per cent. The slag from these furnaces, which amounts to 2.3 per cent of the copper output of the furnace, is sent to the Anaconda smelter for treatment.

**Table X—Results With Various Fuels**

| Fuel Used | Fuel per Ton Copper | B.t.u. per Ton Copper |
|---|---|---|
| Diamondville coal, lb. 12,700 B.t.u. per pound, on grates | 437 | 5,549,900 |
| Diamondville coal, lb. 12,700 B.t.u. per pound, pulverized | 294 | 3,733,800 |
| Fuel oil, lb. 19,180 B.t.u. per pound | 151 | 2,896,180 |
| Natural gas, cu.ft. 925 B.t.u. per cu.ft. | 3,600 | 3,330,000 |

When a furnace has been poled to a point where the refiner considers that the tough pitch stage has been reached, the furnace is tapped and a small test-bar made. This bar is taken immediately to the near-by shop, where a small


ANACONDA REVERBERATORY FURNACES
ANACONDA CONVERTERS
ANACONDA CASTING FURNACES
ANODES
13,650 Tons per Month
890 Tons per Month
12,760 Tons per Month
ANODE CASTING PLANT
Slag 45 Tons per Mo.
Anodes 1,795 Tons per Mo.
Return to Anaconda Reverberatory Furnaces
New Acid
Bluestone Solution from Purification
COMMERCIAL ELECTROLYTIC REFINING TANKS
1,440 Tanks
4,800 Tons Electrolyte
ELECTROLYTIC STARTING SHEET TANKS
90 Tanks - 300 Tons Electrolyte
Anode Scrap 134 Tons per Mo.
Impure Electrolyte 300 Tons per Mo.
Slime 6 Tons per Mo.
Starting Sheets 750 Tons per Mo.
Tank Dirt 15 Tons per Mo.
Anode Scrap 1,800 Tons per Mo.
Slime 96 Tons per Mo.
Cathodes 13,500 Tons per Mo.
Impure Electrolyte 900 Tons per Mo.
13,100 Tons per Mo.
400 Tons per Mo.
Cathodes to Market
REFINING FURNACES
Slag
12,935 Tons
310 Tons per Mo.
Shapes to Market
Return to Anaconda Reverberatory Furnaces
ELECTROLYTIC SLIME PLANT
Settling Tank
Slime Water
Filter Press
Slime Water
Tunnel Dryer
Leaching Tank
Water
Settling Tank
Filter Press
Leach Liquor 600 Tons per Mo.
Slime Water
Slime
Steam Tables
Silver Refinery Perth Amboy N.J. 66 Tons per Month
PURIFICATION PLANT
No.1 Boiling Tanks 1,500 Tons per Mo.
Crystallizing Tanks 750 Tons per Mo.
Impure Mother Liquor 570 Tons per Mo.
Bluestone 150 Tons per Mo.
Water and Steam
100 Tons per Mo.
To Market 50 Tons per Mo.
Bluestone Solution
INSOLUBLE—ANODE TANKS for removal of Cu, As, Sb and Bi
Arsenical Slime 25 Tons per Mo.
Impure Cathodes 15 Tons per Mo.
Impure Electrolyte 485 Tons per Mo.
Anaconda Reverberatory Furnaces
No.2 Boiling Tank
Crystallizing Tanks for removal of Fe and Ni 61 Tons per Mo.
Fe and Ni Slime to Sewer
Restored Acid 60 Tons per Mo.

**Flow sheet of the Great Falls refinery**

section is sawed from the bar and sent to the physical laboratory, where it is polished and an oxygen determination made by the Bardwell microscopic method. The results are telephoned to the refinery and the refiner is advised of the percentage of oxygen in his charge in about twenty minutes after the sample was made and before the first round of warmers has been finished. A wire is made from the same sample bar and the refiner obtains a report on conductivity within 35 or 45 minutes after tapping his furnace. In Table XI is given a typical analysis of the B. & M. brand of electrolytic copper produced at this plant, which shows a copper content of better than 99.95 per cent, oxygen being the chief impurity.

**Table XI—Typical Analysis of B. & M. Copper**

| | Per Cent |
|---|---|
| Copper | 99.955 |
| Oxygen | 0.030 |
| Silver and gold | 0.0034 |
| Sulphur | 0.0019 |
| Iron | 0.0026 |
| Nickel | 0.0014 |
| Arsenic | 0.0015 |
| Antimony | 0.0009 |
| Conductivity annealed wire | 100.50 |

From 80 to 90 per cent of the product of this department is wirebar that is shipped to the Great Falls rod and wire mill, the rest going to the Eastern market.

# Phosphates and Sulphuric Acid

By E. L. LARISON

*Superintendent, Fertilizer and Acid Plants*

AT THE ANACONDA Reduction Works a sulphuric acid and a phosphate fertilizer plant are operated. The sulphuric acid plant consists of three units, two chamber units of 135 tons' and 50 tons' 60 deg. daily capacity, and a packed cell unit of 20 tons' 60 deg. daily capacity.

Sulphur dioxide to supply these plants is drawn from three Wedge roasters treating copper concentrate from Wilfley tables. This concentrate contains about 40 per cent sulphur and 9 per cent copper, and the gas contains 7 to 8 per cent sulphur dioxide. The calcine produced is treated in reverberatory furnaces, so the acid production is a byproduct of the copper reduction process.

The large chamber unit consists of a Glover tower of conventional design, eight large rectangular lead chambers with 29 small cylindrical cooling chambers, and three Gay-Lussac towers. A Glover tower is included in the smaller unit as well as eight rectangular lead chambers and two Gay-Lussac towers. The chambers of both plants are inclosed in substantial brick buildings.

Loss of nitrogen oxides is compensated by a supply of nitric oxide made by oxidation of ammonia. Ammonia liquor is provided to feed this plant.

A plant of 20 tons' 60 deg. daily capacity was erected in 1919 to confirm, on a commercial scale, results obtained in a small system using reaction towers instead of lead chambers. This plant consists of a Glover tower, five brick-packed reaction towers, built in one block, and two Gay-Lussac towers built in one block. The heat of reaction is removed from the reaction towers by circulating 50 deg. acid through them and cooling it in coil-type acid coolers. This plant uses only about one cubic foot of packed volume per pound of sulphur converted per 24 hours.

Sulphuric acid is used at Anaconda for making phosphate fertilizers, treating zinc plant residue to render the zinc content water-soluble, and for leaching sand tailing.

Phosphates of two kinds are made: a concentrated acid phosphate containing about 45 per cent available phosphoric acid, marketed under the trade name Anaconda Treble Superphosphate; and mono-ammonium phosphate containing 13 per cent ammonia and 52 per cent phosphorus pentoxide. Phosphate rock for this operation is mined by the Anaconda company at Conda, in southeastern Idaho. Ammonia is obtained as ammonia liquor from byproduct coke-oven plants situated near Chicago.

The essential steps in the manufacture of Treble Superphosphate are: drying and calcining the rock to remove water and organic matter; grinding and screening; agitation of ground rock with dilute sulphuric acid; separating the solid calcium sulphate formed from the liquid phosphoric acid; concentrating the weak phosphoric acid by evaporation of water; mixing the concentrated acid with ground phosphate rock; curing; and finally, drying and grinding the cured product.

Drying and calcining the rock are done in six-hearth McDougal roasters using coal as fuel. A reduction of weight of about 7 per cent is made in this operation. The calcined rock is ground in ball mills in closed circuit with electrically vibrated screens, to a fineness of 70 to 80 per cent minus 100 mesh. Acidulation of the ground rock is accomplished in Pachuca tanks of acid-resisting construction, with compressed air to perform the agitation. Separation of the solid calcium sulphate from the liquid phosphoric acid is made in a counter-current decantation system consisting of two sets of three Dorr thickeners with diaphragm pumps, all constructed of acid-resisting materials. Overflow from the first thickeners contains about 20 per cent phosphorus pentoxide. The underflow from the third thickener is drawn to a set of Oliver filters, where final removal of phosphoric acid is made by vacuum and washing with water. The cake remaining is then discarded.

The 20 per cent phosphoric acid is concentrated to 45 per cent in vacuum evaporators using byproduct steam for heating. This 45 per cent is mixed, in an open iron pan equipped with mechanical stirring arms, with about one-half its weight of ground phosphate rock. The product contains about 20 per cent moisture. It is cured for several weeks to complete the reactions and is then dried by artificial heat to about 3 per cent moisture. This drying is necessary with this material, but not with ordinary acid phosphate where the water present is consumed in the formation of calcium sulphate crystals not present in Treble Superphosphate. The dried material is ground and screened and is then ready for use.

Ammonium phosphate is made by mixing phosphoric acid and ammonia liquor to form a slurry which is dried in a rotary drier, ground, and screened.

**Mechanical loading of phosphate rock in the mine at Conda, Idaho**

# Smelting Lead at Tooele

By J. O. Elton
*Manager,*
and B. L. Sackett
*Superintendent,*
*International Smelting Company, Tooele, Utah*

J. O. Elton

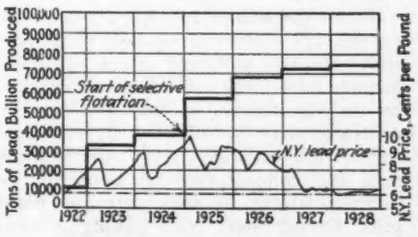
B. L. Sackett

LEAD-SMELTING operations of the Anaconda Copper Mining Company are almost entirely confined to the plant near Tooele, Utah. Since May 27, 1914, when the Anaconda company took over the smelting plant and custom business of the International Smelting & Refining Company at International, Utah, this operation has been continuously under Anaconda's direction, operating as a subsidiary company known as the International Smelting Company. This plant, consisting of a lead-zinc flotation concentrator, a copper smelter, and a lead smelter, is situated near the mouth of Pine Canyon on the western slope of the Oquirrh Mountains, five miles east of Tooele city and about 35 miles southwest of Salt Lake City. This article deals with the lead-smelting operation.

The equipment of the lead smelter consists of five blast furnaces connected to a baghouse, McDougal roasting furnaces for roasting, ten standard Dwight-Lloyd sintering machines for final preparation of the blast-furnace charge, a lead drossing plant, and a lead matte converting plant. For about seven years, from 1914 to 1921, this lead plant was in continuous operation, treating varying tonnages of lead ores and concentrates from the Park City, Tintic, and Bingham districts in Utah, and the Coeur d'Alene district in Idaho, as well as smaller tonnages from miscellaneous camps in Utah, Idaho, and Nevada. During this period the Tooele plant treated about 2,145,000 tons of new lead-bearing charge, from which was produced about 280,000 tons of lead bullion containing 26,700,000 oz. of silver and 110,650 oz. of gold. The charge treated in the blast furnaces during most of this period was quite low in lead (6 to 15 per cent) and relatively high in zinc (5 to 8 per cent.)

For about one year starting in June, 1921, the Tooele lead plant was shut down as a result of a very low metal price and consequent general inactivity in Western lead mines caused by post-war conditions. From July, 1922, when operations were resumed, until now, this plant has been a continuous and growing producer of lead bullion. The steady growth of these operations during the last seven years is well illustrated by the graph in this column, showing lead bullion production, plotted with the New York price of lead.

The graph plainly shows the marked effect of the application of the selective lead-zinc flotation process which was of the utmost importance to Utah's lead-smelting industry when it was introduced, as all of the lead-smelting companies in this district were confronted with the problem of securing an adequate ore supply for their operations. Selective flotation mills made possible the treatment of ores which formerly had been too low grade or too high in zinc to be treated economically by direct smelting; these mills then began to produce a clean, high-grade concentrate for the smelters. The direct effect on the Tooele smelter was a 48 per cent increase in lead production between the years 1924 and 1925. The healthy growth of the effects of the continuation and perfection of the custom flotation business at International is well illustrated in the increase in lead production from this plant from the year 1925, when 57,166 tons of bullion was produced, to the year 1928, when this production almost touched the 75,000-ton mark, an increase of over 30 per cent. It would not be correct, however, to state that all of this growth is attributable to the effect of selective lead-zinc flotation, for a substantial part of it has been caused by other important factors. Among the most important has been the policy of this company, as well as other smelting companies in this district, to aid its shippers in several substantial ways, not the least of which has been the gradual lowering of treatment rates, until the lead-smelting rates in effect in Utah today are the lowest they have ever been. This, of course, has encouraged mining activities and has enabled producing mines to extend both their production and their life by more active ore-development programs. Another important factor has been the active entrance of this company into the field of exploration and mine development in the main producing areas of the state—Tintic, Bingham, and Park City. This alone has added materially to the tonnage of direct-smelting lead ore received at the plant during the last two years.




**Lead bullion production at Tooele plotted with the price of lead**

For a number of years prior to the introduction of selective flotation, the Tooele lead plant treated charges consisting largely of crude lead ores containing relatively high percentages of zinc sulphide. During this period the usual troubles, both metallurgical and economic, arising from the smelting of a high-zinc charge, were encountered. The principal troubles were: (1) Increased furnace delays and shorter campaigns as a result of stack accretions; (2) increased amount of secondaries to be re-treated because of zinc mush in spouts and settlers; (3) increased slag losses because of the type of slag made necessary, and the presence of zinc mush; and (4) increase in the amount of costly iron flux required to produce the type of slag needed, with a consequent reduction in the amount of ore charge smelted.

For several years in this same period the charge treated at Tooele was relatively high in copper (3 to 4 per cent). The main undesirable results of smelting such a charge were: (1) High percentage of coke necessary to obtain ordinarily good reduction; (2) slow furnace speed, caused mainly by the high coke charge used; (3) large matte fall required to care for the copper present, even when running matte up to 25 per cent or more in copper; (4) heat in

upper part of furnace as the result of high coke charge, causing unusually large lead volatilization to baghouse; and (5) large amount of high-copper dross formed in lead wells and in drossing kettles.

The combination of relatively high zinc sulphide and copper on this charge made operations particularly difficult, as certain conditions necessary for the one were deleterious in their effect on the other.

Following the application of selective flotation at the Tooele plant, the zinc content of the charge smelted at the lead plant has, of course, materially decreased and the percentage of lead on the charge has about doubled. Also, as a result of changes in ore supply and methods of operating, the copper content of the lead charge has been almost halved. The result of these changes has tended toward greatly improved and more economical blast-furnace practice. On the other hand, the percentages of fine concentrate, moisture, and sulphur on the lead-plant charge have all greatly increased. The importance and difficulty of a proper preparation of the charge for the blast furnaces on Dwight-Lloyd sintering machines have also increased proportionately. The blast-furnace charge is made up to a larger extent of sinter. Much double sintering has to be resorted to in order to eliminate the sulphur sufficiently and also make a sinter of good quality for the blast furnaces. These factors have been the cause of much trouble in the sintering and roasting operation and have greatly increased costs in this department. Owing to the fineness and high metal content of flotation concentrate handled, the production of very valuable flue dust has increased, making necessary large expenditures for improved dust-collecting equipment, in the form of Cottrell treaters.

The recent ore supply for the Tooele lead plant comes almost entirely from Utah and Nevada. The Tooele concentrator furnishes about 5,000 tons of lead-iron concentrate per month to the lead plant, and the remainder of the ore supply comes from different sources as shown in Table I, each expressed as percentages of the total, exclusive of our own concentrate.

**Table I—Sources of Lead Ore Smelted at Tooele**

| | Per Cent | |
|---|---|---|
| Utah: | | 82.0 |
| Tintic | 48.5 | |
| Park City | 8.5 | |
| Stockton | 18.9 | |
| Bingham | 5.7 | |
| Miscellaneous | 0.4 | |
| Nevada: | | 18.0 |
| Pioche | 16.8 | |
| Miscellaneous | 1.2 | |

Of the total lead-bearing material recently treated at the lead plant about 70 per cent was crude ore and 30 per cent was concentrate. Of the crude ore, about 75 per cent may be classed as "oxide ore" and 25 per cent as "sulphide ore."

Typical of this district, the lead-plant charge contains an excess of silica for proper slag formation, which necessitates the addition of considerable quantities of both iron and lime fluxes. The necessary iron is being furnished in the form of Tooele iron concentrate, generally filtered with the high-grade lead concentrate for metallurgical reasons; and also in the form of scrap iron, charged direct to the blast furnaces. Formerly, iron was furnished in the form of a very low-grade oxidized iron ore, or a barren iron flux. Experience here shows a decided advantage in favor of the use of scrap iron rather than these low-grade or barren iron fluxes. Lime is obtained as a high-grade limerock furnished by the Utah Lime & Stone Company from that company's quarries about 20 miles west of the smelter.

The ore supply for the lead-smelting operation is almost entirely received over the Tooele Valley Railway Company, which connects the plant with the Union Pacific system and with a branch line of the Western Pacific Railway, at Warner Junction, about six miles southwest of International. A small part of the lead-ore supply is shipped from the Utah Delaware mine, at Bingham, over a 4½-mile cable tramway. All crude ores are crushed to the desired degree of fineness and automatically sampled either at the Murray sampler of the Utah Ore Sampling Company or at the International Smelting Company's sampling mill at the plant. All concentrates are thoroughly and carefully sampled at either one of these places. The sampling mill at International is a standard Taylor & Brunton mill, built in two sections, each with a capacity of about 100 tons per hour. One section only of this mill is used for lead ores, the other section being used for copper ores and lead-zinc concentrating ores. The receiving bins at this sampling mill have a total capacity of about 4,000 tons of ore.

From the sampling mill all lead ore rejects are conveyed by belt conveyors to either the sintering plant or blast-furnace bins. These bins are also served by railway tracks, so that ore or concentrates may be dumped directly into them from railway cars. All sulphide ores and concentrates are conveyed or dumped direct to the sintering plant bins, twenty steel bins having a total capacity for about 3,000 tons of fine ore and concentrate. Oxidized lead ores are generally separated into coarse and fines, the coarse being plus 2-in. size. The fine oxidized ore goes to the sintering plant bins and the coarse to the 32 blast-furnace bins, which latter have a total capacity of about 3,700 tons of sinter, ores, and limerock.

Limerock is properly sized at the quarry and is dumped directly into the blast-furnace bins at the smelter. Coke, a product of the ovens of the Columbia Steel Corporation at Ironton, Utah, is dumped from cars on an elevated trestle into a large concrete pit near the lead bins. From this pit the coke is withdrawn through chutes leading to a tunnel below, onto a belt conveyer which elevates it into a cylindrical steel coke bin of 750 tons' capacity that is situated between the lead bins and the blast furnaces.

The various ores and concentrates that go to make up the sintering plant charge are withdrawn in measured quantities from the different bins by Challenge disk feeders and belt feeders onto a main conveyor belt, on which they are bedded. After leaving this bedding belt, the charge is thoroughly mixed in a mechanical mixing device, from which it is discharged to an incline belt conveyor leading to the feed floor of the sintering plant building, where the charge is mechanically distributed to any one of the ten, five-ton feed hoppers above the sintering machines. From each of these hoppers the mixed charge is fed onto the pallets of a sintering machine by means of a swinging-chute feeder.

The ten standard Dwight-Lloyd sintering machines are each 42 in. wide and 264 in. long. Suction for the windbox on each of these machines is furnished by a No. 6 Sirocco fan driven by a 75-hp. motor, which provides a suction of 8 to 10 in. (of water) in the windbox. A charge about 5 in. in depth is spread upon the pallets. Ignition of the sulphide minerals in the charge is accomplished by Salida stoves, in which coke breeze is employed as the fuel.

Double sintering is practiced to a considerable extent at this plant. Four machines are employed in making the first sinter from a charge made up to a large extent of those ores, concentrates, and plant secondaries that are least desirable from a sinter-making standpoint. Each of these first sinter machines handles approximately 160 tons of dry charge per 24 hours, roasting from about 12½ per cent sulphur in the feed to a sinter containing about 5½ per cent sulphur. This first sinter is handled by a long pan conveyor, on which it is moistened and cooled, to a small crushing plant, where it is reduced in size to minus ⅜ in. by screw crushers. From this crushing plant the first sinter is belt-conveyed to certain of the sintering plant charge bins set aside for this purpose, from which it is returned to the machines as part of the charge for final sinter.

The remaining six of the ten sintering machines are used for the production of a final sinter of low sulphur content and good physical quality. The charge to these machines is made up of approximately 60 per cent of the first sinter above mentioned and 40 per cent of the most desirable ores and concentrates. Each of these machines treats about 160 tons of dry charge per 24 hours, reducing the 9 per cent sulphur in the feed to 2½ to 3 per cent in the final sinter, which is dumped from the machines into standard-gage railway cars, in which it is transported to the blast-furnace bins.

With four furnaces in blast, as is usual, the Dwight-Lloyd sintering plant is producing about 570 tons of first sinter and 750 tons of final sinter per 24 hours.

*Copyright by Aerograph Company*

**The Tooele plant of the International Smelting Company**

The gases from this sintering operation pass from the machine fans into a brick flue, from which they go through a humidification chamber 21x27 ft. in cross-section and 110 ft. long. This humidification chamber is lined with Anaconda acid-proof brick laid up in acid-proof mortar. In passing through this humidification chamber at a relatively low velocity, the sintering gases are humidified by high-pressure water sprays so that their relative humidity is about 90 per cent at a temperature of 100 deg. F. as they enter the Cottrell treater, which is connected with the humidification chamber by about 215 ft. of brick flue. This Cottrell treater is of the Tooele flue type, consisting of three parallel units, each containing four electrode sections, placed in series. Each of these effective electrode sections is 10 ft. high, 16 ft. 8 in. wide, and 10 ft. long. The collecting electrodes are made of ¼-in. diameter Duraloy (chrome-iron) rods, suspended vertically in rows which are spaced 8 in. apart across the width of the treater. The rods in each row are spaced at 1⅞-in. centers in a longitudinal direction. Discharge electrodes are ⅜-in. twisted-square rods, suspended horizontally between rows of collecting electrodes and are vertically spaced at 6-in. centers, one from the other. All electrodes are cleaned by air-gun shaking, operated from the top of the treater, augmented by hand-operated shakers for the collecting electrodes. The gases leave the treater, after the solids are well removed, passing to the atmosphere through a 150-ft. acid-proof brick stack. The dust caught by the treater is removed by screw conveyors in the bottoms of steel hoppers underneath the electrode banks, and then loaded into standard railroad cars, in which it is taken to the sintering-plant bins for re-treatment on the first-sinter charge.

A normal four-ton charge to the blast furnaces consists of about 67 per cent of sinter, 16 per cent of coarse oxidized crude ore, 10 per cent of limerock, and 7 per cent of smelter secondaries, mainly coarse converter slag and blast-furnace cleanings. With this charge, about 12 per cent of its weight in coke is used. Scrap iron, from a few pounds up to 500 lb. per charge, is used for iron flux. All of these ingredients, excepting the scrap iron, are weighed out of the blast-furnace bins by means of twenty multiple-beam scale hoppers, into electrically operated, hopper-bottom, charge cars, operating on standard-gage railroad track between the bins and the blast furnaces. The various weighed ingredients of a charge are bedded in the charge car so as to obtain the best possible mixture of the basic, siliceous, and neutral materials, as well as the coke. Whatever scrap iron is required is separately weighed on platform scales in wheelbarrows and placed on top of the charge after it is made up. It is necessary so to handle the scrap iron, as its physical nature prevents it from being handled through the bins from which it could be removed only with considerable difficulty.

The five water-jacketed blast furnaces are each 53 in. wide and 180 in. long at the tuyères. Each furnace is provided with 25 4½-in. diameter tuyères near the bottom of the lower jackets, through which air under about 40-oz. pressure is introduced. The height of the furnace from the center line of the tuyères to the charge floor is about 24 ft. 8 in., of which about 15 ft. is occupied by the ore column. About 275 tons per day is the average duty per furnace.

With four-furnace operation, this plant is producing about 230 tons of lead bullion per day and about 80 tons of lead-copper matte, in addition to the slag. Slag and matte are tapped together from each furnace into a large movable forehearth, in which a separation of the two is made by difference in specific gravity. The matte is tapped from the first forehearth into five-ton ladles placed on railroad trucks, on which it is hauled to the converter building by electric locomotives. The slag overflows from the forehearth into a smaller, second settler and from that into a seven-ton slag pot, in which it is taken to the dump. Lead is removed from the furnace through an Arents siphon tap into three-ton capacity ladles, in which it is carried by monorail crane to the drossing plant near the west end of the blast-furnace plant.

In Table II are given tabulations showing blast-furnace slag and matte analyses for recent practice:

**Table II—Blast-Furnace Slag and Matte Assays**

| | Per Cent | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Cu | Pb | $SiO_2$ | FeO | Fe | CaO | $Al_2O_3$ | S | ZnO | Zn |
| Slag..... | 0.14 | 0.86 | 35.0 | 34.3 | .... | 18.5 | 3.6 | 1.5 | 6.0 | .... |
| Matte..... | 15.25 | 10.9 | 1.0 | .... | 40.8 | .... | .... | 21.3 | .... | 3.4 |

The gas from each blast furnace is drawn off just below the charge floor through a cylindrical dust catcher into the main steel balloon flue, through which the combined gases from all furnaces are carried about 500 ft., by fan suction, into a brick baghouse. This baghouse contains 1,440 cylindrical, woolen bags 18 in. in diameter and 30 ft. long. All solids carried by the gases

**Lead drossing and casting plant**

are caught in these bags, from which they are shaken by reverse air current into pits beneath the bag chambers. From these pits, the fume is removed periodically and either returned to the first-sinter charge or sold to some of the other smelters that have plants in the Salt Lake valley.

The hot matte from the blast-furnace forehearths is taken to the converting plant, where it is poured into 96x150-in. horizontal, barrel-type converters. To these converters, silica is added to flux the iron, during which operation almost all of the lead is blown off in the form of a basic lead-sulphate fume. After the slag is skimmed off, a "white metal" remains, containing about 60 to 65 per cent copper. This "white metal" is transferred from the lead converter into another similar converter containing reverberatory copper matte, where it is blown to blister copper.

The gases from the converter, carrying the lead fume, pass through hoods into a long, steel flue connected to a baghouse. In this steel flue these gases are sufficiently cooled, and their acid content is neutralized, by the addition of fine, dry lime, so that they may be successfully handled in the baghouse. This converter baghouse is of similar construction to the blast-furnace baghouse described in the foregoing, excepting that it is smaller, containing only 960 woolen bags. Lead fume recovered from this baghouse is treated or shipped to other smelters, in the same manner as the blast-furnace baghouse fume.

The high iron slag produced from this converting operation, which contains between 2 and 3 per cent copper, is granulated and treated at the sintering plant, where it is found to be a first-class sinter-making material.

Drossing and casting of furnace lead is done in a new plant put into operation on Feb. 1, 1929. Kettle equipment consists of four 60-ton, cast-steel kettles and two 120-ton kettles. The lead is handled from the blast furnaces to the 60-ton kettles in three-ton ladles by a five-ton, monorail crane. The 120-ton kettles are used for storage, decopperizing, and casting. As a matter of fact, the drossing operation is really the first step in the refining of furnace lead, and its main purpose is the removal of copper.

The heavy dross is separated from the lead in the 60-ton kettles by the usual means of cooling and air agitation. This dross is skimmed by hand from the top of the lead bath into a large dross-basket, with perforated sides and bottom, which is immersed in the lead bath. This basket will hold 6,000 to 7,000 lb. of dross. After being filled with dross this basket is lifted out of the kettle by the monorail crane and held above the kettle until the molten lead has almost completely drained out through the perforations in the basket. The basket containing the dross is then carried outside of the building, where the dross is dumped onto a sloping, elevated platform on which it is cooled with water sprays and then scraped through grizzlies into a railroad car spotted beneath. This dross is quite "dry" (free of metallic lead) and is consequently of such fineness and physical character as to be readily handled through one of the regular blast-furnace bins to the charge cars.

After the heavy dross is removed from the 60-ton kettles, the lead is pumped to one of the 120-ton kettles for storage. After one of these large kettles is filled, and a light dross, which accumulates during the filling process, has been removed, further decopperizing is done by stirring crude sulphur into the lead bath under proper temperature conditions. This final decopperizing operation completes, as nearly as is possible with such a process, the removal of copper from the furnace lead, which was started by the drossing operation. With furnace lead containing about 2½ per cent copper, a cleaned lead is produced containing only 0.001 per cent or less of copper. After the light dross from this final decopperizing operation has been skimmed off, the lead is pumped into steel molds, each of four tons' capacity. After a kettle full of lead has been cast, the blocks are sprayed with water to hasten cooling and solidification, after which they are lifted from the molds with tongs, by an overhead traveling crane, and loaded into flat-bottom, railroad cars.

With this new plant and method for handling lead and dross, it is possible to deliver, dross, decopperize, cast, and load at least 250 tons of lead per day with a crew of fifteen men.

With the loading of the lead blocks into railroad cars, the last step in this lead-smelting operation has been taken. After shipments to East Chicago, Ind., this lead is refined at the plant of another of Anaconda's subsidiaries, the International Lead Refining Company, whose operations are separately described by Mr. Johnson in the following article.

# Western Lead Refined at East Chicago by Parkes Process

By G. E. JOHNSON
*Superintendent, International Lead Refining Company*

BULLION refined by the International Lead Refining Company, at East Chicago, Ind., is received from the International Smelting Company, Tooele, Utah, and the American Smelting & Refining Company, East Helena, Mont. To some extent the refinery is a custom refiner of lead bullion. Lead residues and drosses from all plants of the Anaconda company are treated, and limited quantities of scrap battery plates, lead drosses, and secondary materials are purchased.

The plant is a Parkes process lead refinery of 96,000 tons' annual capacity, producing common desilverized pig lead, antimonial lead, and doré bullion. A description of the plant and operations was presented by G. P. Hulst at the Salt Lake meeting of the American Institute of Mining Engineers, August, 1914. (*Trans.* A.I.M.E., Vol. 49, p. 532, 1915.)

The lead refinery is the original plant, and represents the major operation conducted at East Chicago. The site is an area of 32 acres, 20 acres of which is utilized for the refinery operation; the remainder is divided equally between the Anaconda Zinc Oxide Department and the Anaconda Lead Products Company.

At the refinery the principal operations are performed in one main building of all-steel construction, 480x180x50 ft. Separate buildings are provided for the general office, engineer's office, laboratory, research department, storeroom and shops, change house, and mess hall. The width of the main refinery building is divided into three bays, extending the entire length of the building. Two of the bays are spanned by three fifteen-ton electric cranes, two of 28-ft. span in one bay, and one of 77-ft. span in the other, which travel the length of the building and serve all departments. Three standard-gage railway tracks enter the building on different levels. The general arrangement of the furnaces and kettles permits the flow of most of the metal by gravity from bullion to refined metal.

Gases from the residue and blast furnaces are drawn through brick and steel flues by a 50,000-cu.ft. fan to a baghouse. Gases from the kettles and all other furnaces, with the exception of No. 3 softener, are conducted through brick flues to a rectangular brick stack 4x4 ft., and 100 ft. high.

Electric power for all operations is purchased from the Northern Indiana Public Service Company. Water is obtained from the city mains, which supply a 50,000-gal. tank elevated 50 ft. Water discharged from furnace jackets is returned to a 100,000-gal. sump tank and is pumped to the elevated tank. Steam is supplied from the plant of the Anaconda Lead

Products Company. Fuel oil is received in standard tank cars on the high line track, and unloaded by gravity to storage tanks. Auxiliary equipment is housed in a brick and steel building west of the main refinery building. The building contains two electrically driven air compressors, a centrifugal compressor which furnishes air for oil burners, a 1,000-g.p.m. motor-driven water pump, and a steam-driven fuel oil pump.

Tooele bullion, cast in four-ton slabs, is received in gondola cars; East Helena bullion in 100-lb. bars in box cars. Tooele bullion is unloaded direct from cars by the overhead crane, which delivers the slabs to three 135-ton cast-iron, coal-fired, melting or drossing kettles. The molten bullion is pumped from the kettles by a centrifugal lead pump to the softeners as required.

Three 300-ton softening furnaces are used. They are rectangular reverberatory furnaces fired with coal. The hearth of each furnace consists of a steel box, water-jacketed on the sides and ends, lined with firebrick. The bullion is pumped into the furnace through working doors on either side. In the process of treatment the metal is alternately heated to 1,200 deg. F. and cooled to 900 deg., the resulting skim being removed at intervals of six to eight hours. A charge of 250 tons requires 48-hour softening. Softened lead is drawn off through tap holes to the desilverizing kettles. In addition to the furnace softening operations, approximately one-third of the bullion treated is softened in kettles with hydrated lime, a process developed at this plant in 1925.

Three 135-ton circular cast-iron, coal-fired kettles are used for desilverization. All kettles are served by a fifteen-ton overhead electric crane. Each kettle is provided with a Howard press attached to a jib crane operated by compressed air at 90-lb. pressure. Two additions of zinc are used in the desilverization. Zinc dross produced is delivered to dross storage bins adjacent to the retorts. Desilverized lead, which contains 0.55 per cent zinc, is pumped to the refining furnace.

The refining furnace is a 400-ton capacity coal-fired reverberatory similar in general design to the construction of the softeners. The charge is heated to 1,400 deg. F. Steam used for oxidation of the zinc is then introduced through 1¼-in. iron pipes inserted on each side of the furnace. The dross produced is skimmed from the metal bath until the zinc and traces of antimony and other impurities are removed. Then the metal is cooled to 950 deg. F., and discharged by gravity to the molding furnace, though a separate tap hole is provided to permit molding direct from the refining furnace. Both tap holes are fitted with 2½-in. extra-heavy iron plug cocks. The molding furnace, of 200 tons' capacity, is similar in design to the refining furnace, but not water-jacketed. A 5x5-ft. opening is provided in the roof of the furnace for charging scrap anodes and molding scrap. Adjacent to the furnace are 110 horizontal molds, arranged in an arc for the production of refined pig lead; also, 20 horizontal anode molds for the production of anodes for the Anaconda Lead Products Company. Pig lead and anodes are transferred from the molding circle to scales, weighed, and loaded into cars for shipment.

Six gas-fired Faber du Faur retorts are used for the recovery of zinc from the dross produced by the desilverizing kettles. Each graphite retort bottle is mounted in a separate tilting furnace. The retort is charged with 1,200 lb. of dross, a graphite condenser luted over the mouth of the retort, and the temperature is raised to 2,000 deg. F. As the zinc collects in the condenser, it is tapped out through a hole in the bottom and cast into slabs. When all zinc has been distilled, the condenser is removed, the retort tilted, and the retort bullion cast into bars for treatment in the cupels. Blue powder produced is recharged into the retorts. The zinc recovered is used in the desilvering kettles; and broken bottles and retort clay are charged to the blast furnaces.

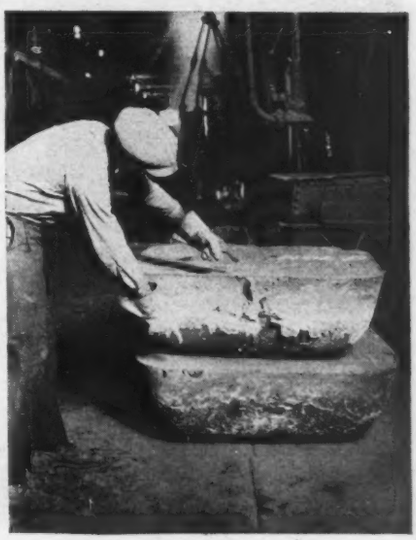
The three Rhodes type cupels are each of the same general size and construction. The "test" is lined with a mixture of crushed firebrick, fireclay, and portland cement. Each furnace is fired with fuel oil. Air for oxidation is introduced through 2-in. blow-pipes with slit nozzles, and retort bullion is charged through doors in the side walls. Litharge is skimmed off by tilting the hearth slightly. Alternate charging of retort bullion and skimming of litharge is continued until the cupel content approximates 90,000 oz. of silver, when the metal is then cleaned by addition of clean litharge and continued blowing with air to oxidize the impurities. The doré bullion, of 995 fineness, is cast into 1,100-oz. bars and shipped to the Raritan Copper Works in New Jersey for parting.

Residue furnace smelting is an intermediate step in the treatment of by-products to separate the antimony, silver, and copper from one another. The furnace is a coal-fired, reverberatory 8 ft. 6 in. x 36 ft. 9 in. x 2 ft. 8 in., similar in general construction to the softening furnaces. Materials charged consist principally of skims from the softening furnaces, litharge from the cupels, and copper dross produced by the removal of excess copper from residue or blast-furnace metal. The charge, including proper proportions of coal for reduction, and galena to form matte, is introduced into the furnace through two circular brick-lined openings in the roof. Products of the operation are a lead bullion containing most of the precious metals, a copper-lead matte, and a lead antimoniate slag low in silver. The lead bullion is pumped to the drossing kettles by a special centrifugal lead pump from a lead well near the firebox end of the furnace; the copper matte is shipped to Tooele, Utah; and the antimony slag is smelted in the blast furnace to produce antimonial lead.

The byproducts treated in the blast furnace consist principally of residue slag and refining furnace skim in the necessary proportions to produce a 15 per cent antimonial lead. The rest of the refining skim is smelted with broken retort bottles, retort clay, furnace brick, and cupel test breakings to product a base bullion. Baghouse dust is recharged.

Blast-furnace equipment consists of two circular, water-jacketed furnaces, 42 in. in diameter at the tuyères, and 14 ft. from tuyère line to charge floor. Each furnace has five 4-in. tuyères. Molten metal flows continuously from the lead well, whereas the slag is tapped at intervals and sent to the dump.

The antimonial metal is transferred to the drossing kettles, drossed with sulphur, and molded into bars for shipment. Base bullion is transferred to drossing kettles, mixed with residue metal, drossed with sulphur and finally refined.



**Some one has attempted to steal these two blocks of lead in transit, as is evidenced by the marks on the side; the weight of four tons each proved discouraging**

# Pioneer Work in Development of White-Lead Manufacture by Sperry Electrolytic Process

By R. G. BOWMAN

*Assistant Superintendent, Anaconda Lead Products Company*

TWENTY PER CENT of the lead consumed in the United States is used in the manufacture of pigments. Of this amount, 80 per cent, or 125,000 tons of lead per year, goes into the production of white lead. The first application of this, the best white pigment known, is lost in antiquity. Ancient methods have characterized its manufacture to the present time, and their quaint clumsiness has been credited with imparting to the product a homespun quality of purity and integrity that could not be secured in any other manner.

The Sperry process for the production of white lead by electrolysis represents perhaps the most striking advance in this industry in recent years. In this process, a bi-fluid cell is employed having a lead anode and an iron cathode, separated by a linen diaphragm. Electrolytes used are sodium acetate solutions containing sodium carbonate. The anolyte contains the exact amount of carbonate necessary to precipitate the lead compound desired; the catholyte contains a relatively large amount of carbonate and acts as a feeder to maintain the exact concentration of the anolyte. The lead dissolves at the anode surface and is immediately reprecipitated as basic carbonate in suspension in the anolyte. A corresponding transfer of $CO_3$ through the diaphragm from the catholyte maintains the concentration of the anolyte unchanged, and the process continues, producing a compound of uniform composition.

Electrolytes are maintained in rapid circulation through the cell. The catholyte in passing through the cell transfers the required amount of $CO_3$ to the anolyte and leaves the cell to pass through a carbon dioxide absorption tower, where it is restored to its original composition. The anolyte circulates without change in composition. White lead leaves the cell continuously in suspension in the anolyte, from which it is removed by settling and filtration, the clear anolyte returning to the cell. The settled white lead is washed, dried, and barreled.

Common desilverized lead is used for the anode; the purity of the white lead produced is independent of the purity of the lead anode, for the reason that the lead alone dissolves, the other metals present in the anode remaining as a closely adherent slime on the anode surface. This is scoured from the anode at short intervals, collected, and treated for the recovery of its metal contents.

THE white-lead plant of the Anaconda Lead Products Company at East Chicago, Ind., was built in 1919; it has a capacity of 28 tons of dry white lead per 24 hours.

**Kettle floor of the East Chicago lead refinery**

Electric power is purchased at 11,000 volts and transformed to the voltages required for power purposes. A motor-generator set supplies the direct current for electrolysis. The substation is designed for completely automatic operation.

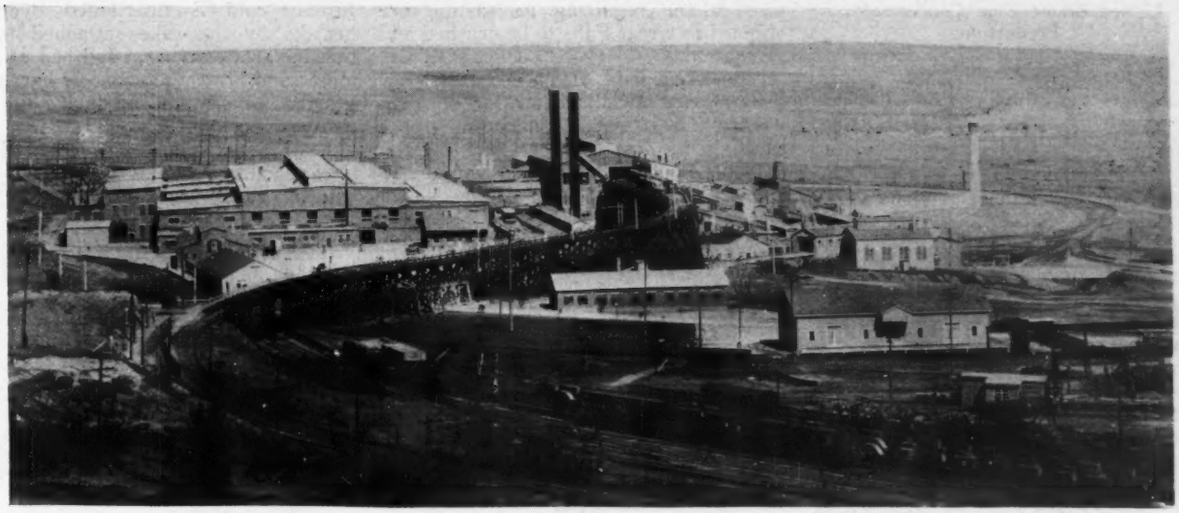
The cell room contains 49 cells of 18 anodes each, with the electrodes connected in parallel and the cells in series. Cell voltage is approximately 3 volts. A rectangular concrete tank with hopper bottom leading to an inverted siphon discharge forms the containing vessel. The anodes are ractangular lead plates 21x30½ in. by 1 in. thick, and weigh 250 lb. each. These are cast with projecting lugs at the upper corners and are suspended in the cell with one lug resting on a knife-edge busbar. Steel sheets, 1/32 in. thick, form the cathodes, each of which is inclosed in an envelope of heavy linen duck supported by a frame of hard rubber.

Catholyte is supplied to each cathode frame by a separate feed tube, circulates about the cathode, and overflows through a spout at the top. The spent catholyte is pumped through a carbonating tower and returns to the cell.

Anolyte enters the cell through submerged feed pipes, circulates about the anodes, and passes out through the bottom of the cell, carrying the white lead in suspension. The combined discharge of all cells, carrying about ½ per cent white lead in suspension, is pumped to a 25-ft. Dorr thickener and the clear overflow returned to the cells. Thickened pulp, containing 30 per cent white lead, is filtered on a Moore filter. Displacement of solution is effected by a series of counter-current washes, the solution being returned to the anolyte system. The cake is washed in hot fresh water until free of soluble salts and discharged. The pulp of white lead and water is pumped to an Oliver filter, which reduces the moisture to 35 per cent before delivery to a Lowden drier having a steam-jacketed hearth, 12 ft. wide and 60 ft. long, of cast aluminum.

After drying, the white lead is finally pulverized in a Raymond mill, delivered to a packing machine, and barreled for market. Its particle size is determined in the original precipitation in the cell, the function of the pulverizing operation being merely to break up small lumps and aggregates produced in drying. The mill is equipped with a heated air supply to insure complete removal of the last traces of moisture from the product before shipment.

The white lead produced by the electrolytic process is a true basic carbonate. It is exceptionally pure, uniform in composition and physical properties, and brilliantly white. It is a superior pigment for all purposes, and finds special application where exceptional whiteness or purity of color is demanded. In its development, Anaconda has written an important chapter in the history of the oldest chemical pigment known to man.



Zinc plant at Great Falls, Mont.

# Electrolytic Zinc Practice at Great Falls and Anaconda

By Albert E. Wiggin and Russel B. Caples

*Respectively, Manager of Reduction Departments in Montana, and General Superintendent of Great Falls Reduction Department*

In February, 1921, a paper was presented at the New York meeting of the A.I.M.E., giving a detailed description of the Great Falls electrolytic zinc plant and a review of the work done prior to that time in the development of process and equipment. ("Electrolytic Zinc Plant of Anaconda Copper Mining Company at Great Falls, Montana," by Laist, Frick, Elton, and Caples. *Trans.* A.I.M.E., Vol. LXIV.) As that paper gave detailed descriptions of equipment used, the data will not be repeated in this paper, and references to the earlier report will refer to it as the "1921 paper." At that time, the Great Falls plant was capable of turning out less than 200 tons of cathode zinc per day. Since 1921, the Great Falls plant has been enlarged to a present capacity of approximately 350 tons of cathode zinc per day, and early in 1928 a plant was completed at Anaconda with a capacity of 175 tons of cathode zinc per day, giving a combined capacity of approximately one million pounds of cathode zinc production per day.

Fundamentally, the process and equipment in use today are the same as described in 1921, but each year has brought metallurgical and mechanical improvements, resulting in improved recoveries and lowered operating costs. Radical changes in composition of feed, through changes in concentration practice, and widening of the field from which concentrate supply is drawn, have brought new problems. That these problems have been successfully worked out is evidenced by the fact that plant capacity has been steadily increased, and the highest ampere efficiency ever obtained by both plants was reached in March, 1929. The record of these plants is ample proof of the stability and flexibility of the electrolytic zinc process, employing low acid strength and low current density, and definitely removes it from classification as an experiment or as a process designed for treatment of a limited class of zinc ores.

The original Great Falls plant was built primarily to treat Anaconda company zinc ores from the Butte district, custom ore business being considered a minor matter. Development of selective flotation completely changed the picture by making available large tonnages of high-grade zinc concentrate in Utah, Colorado, and Idaho, in addition to lengthening the life of Montana mines, and has kept the increasing plant capacity largely filled with custom zinc concentrate. Most of the custom concentrate is considered as a byproduct of lead operations, but in some instances, notably in Butte, lead concentrate is a byproduct of zinc production. The combination of selective flotation and a good market for zinc concentrate has greatly stimulated development of complex ores in Colorado, Utah, Montana, and Idaho. The present supply of zinc for the Anaconda company plants originates as follows: 50 per cent in Montana, 33 per cent in Colorado and Utah, and 17 per cent in Idaho and from miscellaneous sources. These percentages are approximate and are continually changing with variations in metal prices and grade of ore, and are also subject to seasonal variation and changes in supply of miners in the different districts.

The effect of some of these changes on analysis of concentrate treated is shown in the table below. In addition to the analyses given, analyses for arsenic, antimony, cobalt, nickel, cadmium, and other impurities have undergone marked changes with change in grade of concentrate produced and are markedly different for different districts.

| Year | Per Cent Zn | Per Cent Pb | Per Cent Cu | Per Cent Fe |
|---|---|---|---|---|
| 1920 | 37.3 | 7.5 | 1.58 | 11.5 |
| 1923 | 49.5 | 7.0 | 0.85 | 6.0 |
| 1925 | 52.0 | 4.5 | 0.70 | 6.0 |
| 1928 | 53.0 | 3.65 | 0.55 | 5.0 |

The production of zinc concentrate in the United States outside of the Tri-State and Wisconsin districts, as reported by American Bureau of Metal Statistics, is shown in the table at the top of the next page.

**United States Zinc Concentrate Production**
*(Exclusive of Tri-State and Wisconsin fields)*

| Year | Tons | Per Cent Zn | Per Cent Pb |
|---|---|---|---|
| 1921 | 38,433 | 39.84 | 11.60 |
| 1922 | 82,121 | 45.20 | 7.43 |
| 1923 | 169,496 | 49.42 | 6.60 |
| 1924 | 227,955 | 47.82 | 6.30 |
| 1925 | 253,989 | 51.42 | 4.36 |
| 1926 | 404,485 | 51.63 | 4.13 |
| 1927 | 500,675 | 53.07 | 2.46 |
| 1928 | 551,187 | 53.43 | 3.13 |

This shows clearly the effect of the development of selective flotation on the tonnage and grade of the zinc concentrate produced, most of which comes from the Rocky Mountain region.

In addition to concentrate, baghouse fume from East Helena is treated in the electrolytic zinc plant at Great Falls. Practically all of the concentrate treated is received at the plants in open-top, hopper-bottom cars. It is all flotation concentrate, containing an average of 9 per cent moisture. During a large part of the year the concentrate is received frozen in the cars, requiring thawing before sampling and unloading. Sampling is done by boring holes through the concentrate, using a long auger, and combining the borings from all holes in each car as the sample from that car. The number of holes drilled in each car depends upon the length of the car and varies from 27 to 40. Remarkably close checks may be obtained with this method of sampling. Any number of cars, as agreed upon between shipper and buyer, are combined to make a sample lot. After sampling, cars are dumped into roaster feed storage bins, using compressed air to loosen the concentrate so that it will fall through the hoppers. When hoppers are packed properly, loss of concentrate in transit is negligible even with concentrate containing from 10 to 12 per cent moisture, but frequently cars are received with water dripping from hoppers, the packing of burlap or rags acting as a filter.

Uusually from ten to fifteen mills are shipping concentrate to these plants, each product being different from the others in analysis and with widely varying tonnage. At Great Falls, ample bin capacity is available from the old copper-plant operations to permit blending the various concentrates to give a uniform roaster feed. At Anaconda, practically no bin storage exists ahead of the roasters, making it necessary to take a full carload of concentrate of each variety as dumped.

Each plant has fourteen 25-ft. shell diameter, seven-hearth, Anaconda-Wedge roasting furnaces. All of the Great Falls furnaces, and whatever additional furnaces at Anaconda are needed, are operated to supply the Great Falls plant. As all of the Anaconda furnace calcine goes through the same cooling and screening plant, the product is a composite of the production of all furnaces and a portion of this product is shipped to Great Falls. Open-top steel cars, having a sealing device in the hoppers to avoid dust leakage in transit, are used for this service, and transit losses are small—chiefly from wind. All Anaconda furnace barrings and oversize calcine requiring re-roasting are shipped to Great Falls to be crushed and re-roasted.

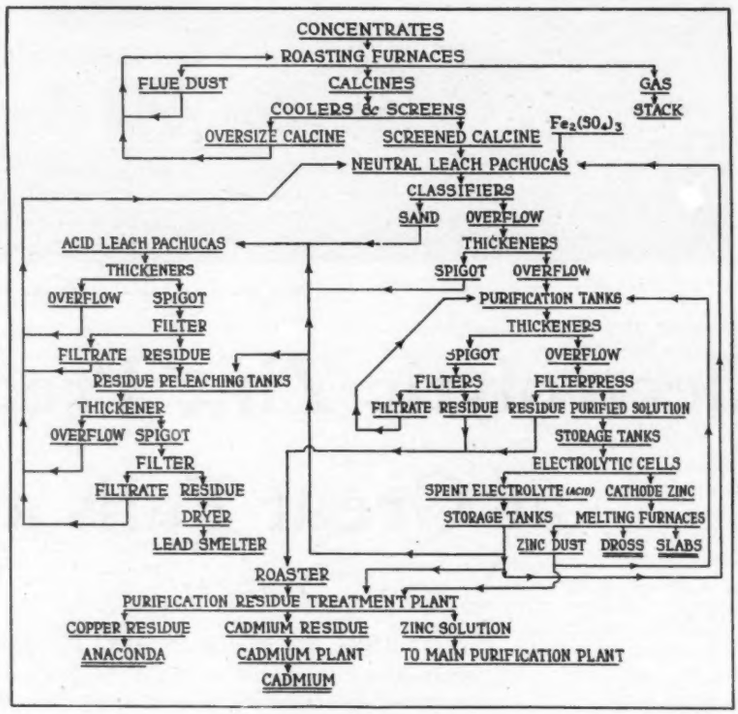
The Anaconda furnace gases pass through a Cottrell treater for dust recovery. Great Falls furnace gases pass through the dust chamber built for the old copper-plant operations, where gas velocity is reduced to a point that allows most of the dust to settle; from this the settled dust is drawn through hoppers into cars for return to the roasting furnaces. After leaving the dust chamber, the gases pass through a long flue leading to the stack, where some additional dust settlement takes place. The dust recovery at Great Falls compares very favorably with recovery at Anaconda. Total flue dust re-treated, including circulating load, averages a little over 15 per cent of the weight of new material treated.

From 35 to 40 tons of new feed, in addition to flue dust and calcine barrings, is roasted per furnace day, producing calcine containing 0.2 per cent sulphur as sulphides. The calcine is cooled in rotary coolers having trommel screens fastened on the discharge end. All calcine fed to the leaching plant passes a $\frac{1}{8}$-in. mesh screen. Oversize from this screen and undersize of a $\frac{3}{8}$-in. screen is crushed through rolls and returned to coolers to be re-screened. Oversize of the $\frac{3}{8}$-in. screen is crushed and returned to the roasters on account of its high sulphide sulphur content.

At the end of each two-year period of operation the furnaces are shut down in rotation for re-sanding of hearths and general overhauling. Hearth accretions are broken up and removed from the furnaces, crushed and returned for re-roasting. Fresh sand is fed to the furnace and is distributed over the hearths by the rakes attached to the furnace arms, giving a full bed of fresh loose sand. Little plowing or barring of hearths is then required during the next two-year period.

Anaconda furnaces are oil-fired and Great Falls furnaces are fired with natural gas. In each instance, two burners, spaced 180 deg. apart, are fired directly onto the seventh hearth, eliminating the use of fireboxes. When coal was used, either lump or pulverized, outside fireboxes were used. When firing with pulverized coal of 9,500 B.t.u. heat value, approximately one ton of coal was required for seven tons of new feed, whereas approximately 14 gal. of oil, with 18,500 B.t.u. heat value and specific gravity 0.92, is used per ton of new feed. With natural gas, having a heat value of 930 B.t.u., net, approximately 2,400 cu.ft. is required per ton of new feed. Maximum temperatures of about 750 deg. C. are obtained on the third and fourth roasting hearths. Calcine is discharged at about 500 deg. C., and gases leave the furnace at about 525 deg. C. The volume of gas per furnace at the uptake varies from 5,500 to 6,000 cu.ft. per minute at 0 deg. C. and 760 mm. mercury, and contains 3 per cent sulphur dioxide by volume. The furnace arms are water-cooled, requiring approximately 140,000 gal. of water per furnace per day.

In the roasting of concentrate containing 5 per cent iron or less, practically all of the iron combines with zinc as $ZnO.Fe_2O_3$, which is nearly insoluble in dilute sulphuric acid. As the iron content increases above 5 per cent, the combination of zinc and iron oxides can be partly controlled by careful regula-




**Flow sheet of the Great Falls plant**

tion of temperatures on the various hearths.

The total sulphur content of the calcine produced is regulated by the acid requirements of the plant, sulphide sulphur content being kept at all times at 0.2 per cent or less. Acid losses are small, being chiefly as lead sulphate, basic iron sulphate, and entrained zinc sulphate in the residues discarded from the plant.

Actual leaching operation is divided into three steps: 1, Neutral, or purification, leach; 2, acid, or finishing, leach; and 3, re-treatment of residue from (2).

The first two leaching steps are continuous leaching operations carried on in unlined Pachuca tanks using 30-lb. compressed air for agitation. Residue leaching is a batch leach operation conducted in unlined, mechanically agitated wooden tanks. No lead-lined tanks are used for leaching and no outside heat is supplied for leaching in the first and second steps. Solution going into the residue re-leaching operation at Great Falls is heated before adding the residue. Elevators equipped with bronze buckets are used for elevating pulp.

As less than 5 per cent of the iron contained in the calcine is soluble in the acid strength employed in the first and second leaching steps, scrap iron is dissolved in spent electrolyte and the resulting ferrous sulphate solution is oxidized to ferric sulphate by manganese dioxide and is added to the first, or neutral, leach step, as required for purification of solution for arsenic and antimony.

To the neutral leach is added all of the calcine, but only a part of the acid (spent electrolyte), resulting in the presence of an excess of zinc oxide for precipitation of such impurities as iron, silica, alumina, arsenic, and antimony. The leaching solution is a mixture of spent electrolyte, acid thickener overflow, and returned solution from re-leaching of the residue, to which is added sufficient ferric sulphate solution to precipitate arsenic and antimony. The pulp from this leach goes to classifiers and thickeners, the thickener overflow going to purification tanks and the spigot product to the second, or acid, leach. Classifier sand goes to the acid leach tanks.

Spent electrolyte is added to the second, or acid, leach in sufficient quantity to dissolve all uncombined zinc oxide in the residue from the first, or neutral, leach and to insure an excess of acid in the leach discharge, excess acid being provided to insure complete solution of free zinc oxide. The discharge from this leach goes to acid thickeners, the overflow being returned to the first or neutral leach and the spigot product delivered to filters.

At Great Falls, the undried filter cake is sent to mechanically agitated leaching tanks, previously filled with hot, spent electrolyte. As the leach progresses, part of the zinc-iron compound is decomposed, giving a solution rich in ferric sulphate and lowered in free acid. A further reaction between the ferric sulphate solution and zinc ferrite results in solution of the zinc oxide and precipitation of the iron as basic sulphate. This leach is discharged, while still acid, to thickeners, the overflow going to the first, or neutral, leach and the spigot to a Moore filter, where the residue is washed nearly free of entrained zinc sulphate solution. The pulp resulting from Moore filter operation goes to Oliver filters for dewatering and the Oliver cake goes to driers for partial drying before shipping to a lead smelter for recovery of lead, silver, gold, and copper.

Residue from the final filters contains from 35 to 40 per cent moisture and is dried to about 15 per cent before shipment to the lead smelter. For this drying operation, a direct-fired, rotary drier, 9x60 ft., brick lined, is used. Dried residue is discharged from drier to drag conveyor, which discharges into railroad cars.

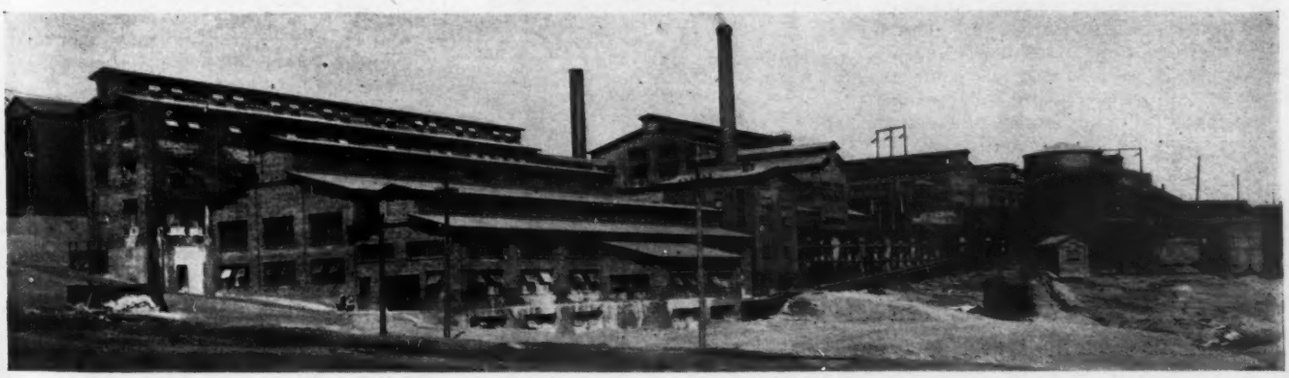
At Anaconda, residue from the second, or acid, leach is filtered on Oliver filters and is dried in a 9x60-ft. rotary direct-fired drier to about 10 per cent moisture. This partly dried residue is mixed with concentrated sulphuric acid in a pug mill, and the resulting mud is fed into an oil-fired McDougal roasting furnace. The calcine from this operation is treated with an additional quantity of concentrated acid and is put through a second McDougal furnace, with a finishing temperature below the decomposition temperature of zinc sulphate. Acid addition and roasting are carried out in two steps on account of the impossibility of adding all of the acid at once without producing a pulp too thin to handle satisfactorily. The amount of acid added is regulated by the zinc content of the residue treated and the finishing temperature of the second roast is regulated by the amount of soluble iron desirable to produce in the finished calcine. This calcine is returned to the leaching plant and agitated with water to extract the soluble zinc, iron, and copper sulphates, the resulting pulp being sent to a thickener. Thickener overflow goes into the main solution system for purification, and the thickener spigot product goes to a Moore filter for washing; the Moore filter pulp goes to an Oliver filter for dewatering. The Oliver cake goes to a drier, and the dried product is shipped to a lead smelter.

Each method of residue re-treatment, that at Anaconda and that at Great Falls, decomposes a part of the zinc-iron compound formed in roasting zinc concentrate, and each method is probably the best under the set of conditions existing at that plant. The Anaconda method requires cheap sulphuric acid, obtainable at Anaconda, because an amount of acid equal to that used in residue re-treatment and returned to the leaching plant must be discarded from the process to avoid building up the zinc content of solution.

The overflow from the neutral thickeners following the first, or neutral, leach contains copper and cadmium, which must be removed prior to electrolysis. This is accomplished by agitating the solution with atomized metallic zinc, or zinc dust. To effect complete removal of cadmium, a large excess of zinc dust is required, resulting in production of a purification residue high in zinc and requiring re-treatment for recovery of its zinc content. The amount of zinc dust required varies from 3.5 to 4.5 per cent of the weight of zinc produced, the amount depending largely upon the copper content of the solution to be purified.

Solution purification is carried out in batches in mechanically agitated tanks, each batch being tested for copper and cadmium before being discharged to a thickener. Thickener overflow is filtered through Shriver presses, filtered solution being pumped to storage tanks for the electrolyzing division. The thickener spigot product is sent to an American filter, the filter cake and Shriver resi-

**The newer zinc plant, at Anaconda**



Albert E. Wiggin

due, containing copper, cadmium, and zinc, being sent to the purification residue re-treatment plant for recovery of the three metals.

The residue resulting from copper-cadmium purification contains metallic zinc, copper, and cadmium, in addition to any slime contained in the neutral thickener overflow. To convert the metallics into oxides, the residue is subjected to a low temperature roast.

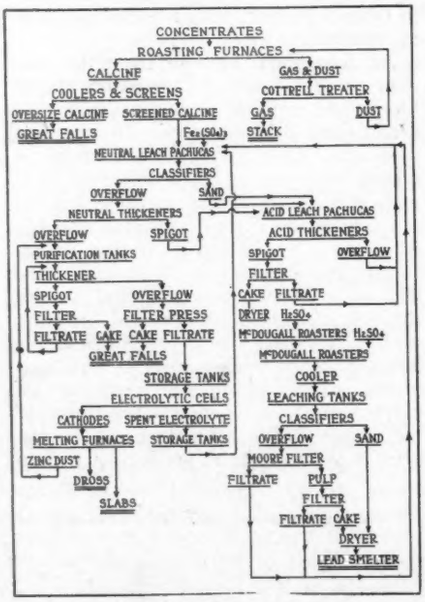
Roasted residue is leached with spent electrolyte from the zinc electrolyzing division, adding sufficient residue completely to neutralize all free acid. Most of the copper, cadmium, and zinc are extracted in this leach, giving a solution high in copper and cadmium. The leaches are discharged to a thickener, the overflow going to a mechanically agitated tank, where most of the copper is precipitated on metallic zinc, giving a high-grade copper precipitate and leaving a solution of zinc and cadmium sulphates. This solution, after removing the copper residue, is treated with additional metallic zinc to remove most of the cadmium as a high-grade cadmium product, which becomes the feed for the electrolytic cadmium plant. The solution, purified of copper and cadmium, is returned to the main zinc leaching plant for final cadmium purification. The residue resulting from leaching of the purification residue, and the high-grade copper precipitate, are shipped to the Anaconda copper plant. Purification residue from the Anaconda plant is shipped to Great Falls for re-treatment, avoiding duplicating this small auxiliary department.

Only a small part of the zinc used in purification of solution is finally lost in the copper residues sent to Anaconda for smelting, and most of the cadmium dissolved from the roasted concentrate is recovered as a high-grade cadmium residue. Fractional precipitation of copper and cadmium from zinc sulphate solution rich in both impurities may be carried out very closely, as freshly precipitated cadmium is immediately available for copper precipitation.

The high-grade cadmium residue obtained from purification residue re-treatment is dissolved in sulphuric acid and the resulting solution is purified of the last traces of copper and filtered, giving a solution rich in cadmium and zinc. This solution is then electrolyzed, using lead anodes and aluminum cathodes, keeping the cell voltage under the decomposition voltage of zinc sulphate to avoid deposition of zinc with cadmium. The cadmium deposit is stripped from the cathode blank and melted under caustic or oil, and then cast into shapes for market. Formerly, practically all cadmium was marketed in pencil-like form, but in recent years the uses of cadmium for electroplating have increased so rapidly that a large part of present production is marketed in various anode forms.

Very high-grade cadmium is produced as a byproduct of electrolytic zinc operations, and production from such sources is now a major factor in the cadmium market, though no production from these sources was made ten years ago. Anaconda company production in 1928 amounted to 891,000 lb.

As successful electrolytic deposition of zinc from a sulphate solution is primarily dependent upon the purity of the solution fed to the cells, every effort is made in the leaching and purification plants to produce a solution entirely free from metals electro-negative to zinc, and materials entering into electrode and tank construction are carefully selected to avoid contamination from these sources. Although a great amount of study has been put on the effect of a number of impurities on the electro-deposition of zinc from a sulphate solution, and fairly accurate limits




Flow sheet of the Anaconda electrolytic zinc plant

have been established for single impurities, comparatively little is known regarding the combined effect of these impurities in varying proportions. Also, a great deal of effort has been spent in an attempt to develop a suitable addition agent to lessen the effect of certain impurities, and some success has been had for a given set of conditions, but when

Russel B. Caples

a new variable is introduced it immediately establishes a new set of conditions and requires a new form of treatment. Given a pure solution and pure materials in contact with solution, high ampere efficiency is easily and consistently obtained, but with either impure solution or impure materials erratic results are obtained; therefore, the highest possible standard of purity is maintained, and occasional small amounts of impurities which may get by the established precautions have very much less effect than if addition agents were relied upon.

The Great Falls electrolyzing division is divided into eight electrical circuits, each containing 144 cells, arranged in cascades of six cells each. In addition there are twelve cells connected so that they may be cut in on any circuit to replace those taken out of service for cleaning. This gives a total of 1,164 cells of which 1,152 are continuously in service. Each circuit, or unit, is supplied with current from a rotary converter having a capacity of 10,000 amperes at 580 volts. The Anaconda plant has four such units, each connected to a motor-generator set of the same capacity as the Great Falls rotary converters. Motor-generator sets were installed at Anaconda on account of their greater stability under the electrical conditions existing there, this factor being considered sufficient to offset their small loss in conversion efficiency compared with rotary converters.

No material change in the construction of the cells and electrodes has been made since the details were published in the 1921 paper. Spacing of electrodes is being changed from 4 in. from center to center of anodes, to 3-in. centers, resulting in a material reduction in power requirements. Current density remains at approximately 30 amperes per square foot of cathode area.

The solution is distributed from storage tanks through lead pipe lines to individual cascades of cells and is taken from header lines through iron pipes to the individual cells, flow to each cell being regulated so as to maintain practically a constant acid and zinc concentration in each cell. Acid strength is determined by measurement of the electrical resistance of the electrolyte, readings so obtained being occasionally checked against a standard chemical

method. The feed to the cells averages about 110 grams of zinc per liter, and the cell discharge averages about 105 grams of sulphuric acid per liter. Spent electrolyte is collected by a system of launders and delivered to storage tanks, from which solution is drawn as needed for the leaching plant and purification residue re-treatment plant.

Each cell is cooled by circulating water through a lead coil placed in the cell. Cell temperatures vary from 40 to 50 deg. C., depending upon atmospheric and cooling-water temperatures. The amount of water used for cooling purposes varies from 8,000 to 15,000 gal. per ton of cathodes produced.

Cathodes are removed and the zinc deposit is stripped at 24-hour intervals. For several years the stripping interval was 48 hours. The change from the 48 to a 24-hour interval has been an important factor in increasing and stabilizing ampere efficiency; the resulting saving in power and increased production more than offsets the increased labor cost. With absolutely pure solution, little improvement in ampere efficiency is gained through establishing the shorter stripping interval, but the sensitiveness of the zinc cell to the effect of impurities in solution increases rapidly with the length of the period of deposition, and a solution of sufficient purity to give high ampere efficiency for 24 hours may be sufficiently impure to give a very low efficiency at end of a 48-hour period. Shortening the period of deposition makes the cell much more resistant to the effect of occasional temporary imperfections in purification of solution and brings about a much more rapid recovery from the effects of an abnormal amount of impurity. With a 24-hour stripping interval, changes in ampere efficiency are normally very gradual, and the stability of cell operation has been greatly improved.

One man removes cathodes, strips the deposit, brightens contacts, and replaces the cathodes from twelve cells each day, against nine per day with the 48-hour stripping interval. He also delivers his load of zinc to the scales for weighing and then to the melting furnaces. Cathode production averages about 7,000 lb. of zinc per stripper per day.

Each group of cells is taken out of service at six months' intervals for a thorough overhauling. The sludge deposit is removed, the cell repainted, insulation is renewed, anodes are cleaned and straightened, and any other needed repairs are made. No ill effect has been noted from the mist given off from the cells, either on the health of the workmen, some of whom have been continuously employed as strippers for fourteen years, or on the building steelwork. The original I-beams carrying the trolleys and hoists used in stripping, and placed directly over the cells, are still in service and in good condition after nearly thirteen years of steady service.

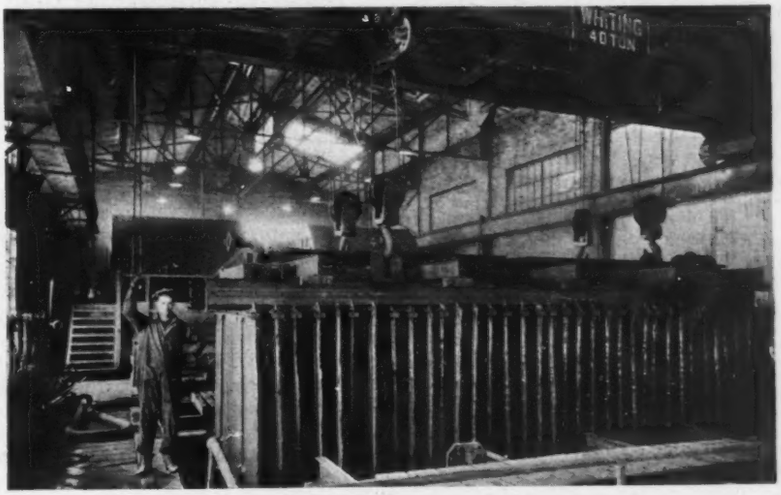
The zinc cathode sheets from the electrolyzing division are stacked as stripped in piles of approximately one ton each on small cars; these cars, carrying four piles each, are delivered to the melting and casting department, where each pile is lifted by an electric hoist, using an iron fork slipped under the pile, to an inclined charging platform above the furnace. A door in the roof of the furnace is opened and the pile of cathodes slides into the furnace. Charging, melting, and casting are carried on simultaneously and continuously for sixteen hours each day. At the end of this period, the furnace is cooled down, dross is skimmed, and the furnace is again brought up to melting temperature. Approximately 150 tons of molten zinc is constantly kept in each furnace, so a considerable part of the melting of fresh cathodes is done under the surface of the zinc bath.

Three furnaces are installed at Great Falls and two at Anaconda, each furnace having a normal melting capacity of 100 tons of cathode zinc per day. Those at Great Falls are gas-fired; those at Anaconda are oil-fired. Coal-firing was used at Great Falls for several years before either oil or gas was available. The character of fuel makes little difference in dross production, the quality of cathodes and rate of melting having a much greater effect. A certain minimum amount of dross seems to be formed per square foot of bath area, independent of the tonnage treated, so it is advisable to keep the tonnage up to the maximum that can be melted in each furnace without overheating. From 3.5 to 4.5 per cent of the zinc in the cathodes melted goes into dross. A drop in ampere efficiency, insufficient in severity to cause a marked difference in appearance of the zinc cathode sheet, will nearly invariably be reflected in an increase in the amount of dross produced. If the furnaces are not overcrowded, the volatilization loss from the furnaces and from reworking dross will not exceed 0.25 per cent of the cathodes melted.

Dross, as formed in the furnaces, is a pasty mass high in metallic zinc. Some ammonium chloride is worked into the dross in the furnace, freeing much of the metallic zinc and generating heat. The partially worked dross is then skimmed from the furnaces through side doors into concrete buggies and transferred to heated, revolving drums, where more ammonium chloride is added and the drum is revolved until metallics and dry dross are separated. Normal consumption of ammonium chloride for both furnace and drum treatment of dross is slightly less than

**Casting electrolytic zinc at Anaconda**

**Moore filter installation at the Anaconda plant**



1 lb. per ton of cathodes melted. The metallics, in molten condition, are drained from the drum and returned to the melting furnaces.

After removing all metallics, the resulting dross is either sold to plants desiring a high-grade zinc material of high purity, or is re-treated through the roasting and leaching plants at Great Falls. During periods when concentrate supply is inadequate to keep both plants operating at capacity, dross is retreated; when an ample supply of concentrate is available, dross is sold.

Each furnace is equipped with two casting wells, surrounded by muffles to prevent air leakage into the furnaces, from which zinc is dipped by hand ladles suspended from trolleys running on overhead I-beams, and the zinc then cast into stationary molds arranged in rows parallel to the long axis of the furnace. As soon as the zinc slabs have solidified they are stamped with a lot number for identification, and the molds are dumped and replaced for the next pouring. Water sprays are placed underneath the molds for cooling. The slabs of zinc are stacked in rows until the day shift, when they are check-weighed and loaded for shipment. Shot samples are taken during the casting period. These samples are made up to represent each day's production from each furnace, to correspond with the lot number stamped on the zinc slabs, and are analyzed for lead, cadmium, and iron.

One man can dip from the furnace and cast into slabs 30 tons of zinc in an eight-hour shift. Various schemes to improve upon this method of casting zinc have proved unsuccessful. For instance, a casting wheel, with the molds placed on the periphery of the wheel, was installed. The cooled slabs were removed from the molds by a vacuum lift and stacked at one side of the wheel. No appreciable money saving was made and the appearance of the slabs was damaged by the movement of the wheel before the zinc had solidified. As the slabs weigh only 50 lb. each, and the rate of casting is limited by the rate of melting, it is improbable that machine casting will replace hand casting until a system of melting is devised whereby a large tonnage of zinc can be cast in a comparatively short period of time.

An electric furnace, heated by a resistor of crushed carbon placed in a trough along the sides of the furnace, was tried in an attempt to reduce dross production, but was not successful either in reducing the amount of dross or the cost of melting.

For making zinc dust, molten zinc is transferred from the melting furnaces to graphite crucibles having carbon rods, tapped with ⅛-in. holes, placed in the bottoms. As the molten zinc falls from one of these openings it is caught in a jet of compressed air from an atomizing nozzle and is blown into a settling chamber. Air from this chamber is exhausted through bags to avoid dust loss. The atomized zinc, or zinc dust, is removed into cars to be transported to the purification division for use in removal of copper and cadmium from solution.

The Great Falls zinc plant substation receives power at 110,000 volts. This is stepped down through nine 6,100-kva., 3-phase, 60-cycle, oil-insulated, water-cooled transformers to 406 volts. Each transformer supplies one 5,800-kw. synchronous rotary converter capable of delivering 10,000 amp. at 580 volts direct current. Each transformer-converter unit supplies 144 cells in the electrolyzing division, one unit being a spare to be used during repair and cleaning periods. In addition to the complete spare unit, a spare transformer core and a spare rotary-converter armature are kept for emergency use. In addition to the transformer-converter units, two banks of transformers supply 2,200-volt power for the leaching plant, air compressor, and other auxiliary purposes.

The substation is supplied with cool clean air from two air-washing plants, each of 100,000 cu.ft. per minute capacity. Heated air and dust from rotary converters are exhausted by fans placed in wall openings above.

The zinc plant at Anaconda receives power from the Montana Power Company's system over a 100,000-volt, 3-phase, 60-cycle transmission line to an outdoor high tension bus. From this bus, with the necessary disconnects, four 100,000-volt outdoor oil circuit breakers control the power to four outdoor transformers, which reduce the 100,000 volts to 6,600 volts, 3-phase, 60 cycles, supplying four 5,000-kw. motor-generator sets.

Each motor-generator set comprises a 7,500-hp., 6,600-volt, 3-phase, 60-cycle motor, driving two 2,500-kw., 560-volt, direct-current generators, which, in turn, supply, through the necessary circuit breakers and auxiliaries, direct current to their individual cell circuits. To condition the air properly, each motor-generator unit is equipped with a 15,000-cu.ft. per minute humidifier.

All auxiliary power is supplied from the reduction plant substation over a 2,200-volt, 3-phase, 60-cycle line, being transformed to 440 volts for motors, and 220 or 110 volts for lighting throughout the plant.

*Steam*—Steam is used chiefly for the heating of buildings. At Great Falls some steam is also required for heating the spent electrolyte used in residue leaching. Small quantities of steam are also required for heating oil and miscellaneous purposes, and, during the winter, thawing of concentrate requires a fairly large quantity. As climatic conditions and building construction determine, to a great extent, the amount of steam required, no exact figures on requirements per ton of zinc or other unit will be given.

*Air*—Low-pressure air is used for leaching tank and filter agitation, the pressure varying from 20 to 30 lb. per square inch. High-pressure (90 lb.) air is used for zinc-dust manufacture, for unloading concentrate cars, for drawing concentrate from bins, for emergency tank agitation, and for various minor uses, such as operation of drills and riveting hammers.

*Water*—In addition to cooling the roasting furnaces and electrolyzing cells, water is required for the cooling of transformers and compressors, for pumps, for general plant clean-up, for washing filter residues, for cooling casting molds, and other miscellaneous purposes. As ample water supply is available at Anaconda and at Great Falls, no serious attempt is made to conserve the water supply. Total water consumption for all purposes varies from 18,000 to 27,000 gal. per ton of zinc, depending upon seasonal variations in temperature.

*Power*—At Great Falls, nearly 95 per cent of the total power required for zinc-plant operations is used by the electrolyzing division; 3.5 per cent is used by the roasting, leaching, drying, and purification residue treatment divisions; and 1.5 per cent is used for pumping, tramming, shops, and miscellaneous purposes. The greater the output of zinc, the higher the percentage of total power requirements used by the electrolyzing division, as the power required for electrolysis varies directly with the quantity of zinc produced (assuming no variation in current density, in acidity of electrolyte, or in ampere efficiency), whereas auxiliary power requirements do not increase in proportion to the tonnage of zinc produced.

*Labor*—In general, a higher type of labor is required for electrolytic zinc work than for the usual smelting and refining work, this being especially true in the leaching and purification divisions, where a careful control of operations must be maintained to insure high precipitation efficiency. Working conditions compare favorably with conditions in other reduction works, and no trouble is experienced in getting and retaining a desirable class of employees.

*Sampling and Assaying*—Some of the plant control analyses, such as acid, iron, and copper determinations on solutions, are made in the plant by the operators, but the greater part of such work is done in the general plant laboratories, where many analytical methods have been developed to give greater speed and accuracy for this particular class of work, enabling the operators to obtain reliable analyses sufficiently rapidly to be used as a guide or control for plant operation. Research laboratories are maintained for the development of improvements in the process and for carrying out test work on new concentrate supplies, in addition to carrying on the necessary routine testing work.

We wish to take this opportunity of expressing our appreciation of the splendid work done by the men in active charge of the electrolytic zinc operations for several years past and by those in charge of auxiliary and service organizations. The records that have been established for production and for cost of production are largely the result of their consistent effort and co-operation, and we submit this brief description of present operations as being primarily a record of their work.

# Fuming Off Zinc From Lead Slags at East Helena

By Alexander Laist

EXPERIMENTS in recovering waste zinc from lead-furnace slags were begun late in 1926 at Tooele, Utah. The first work was done in a copper converter, and in fact the furnace now used may be described as a non-tipping converter.

The design of a suitable plant for East Helena, Mont., was begun in June, 1927. This plant now consists, in the first place, of a water-jacketed furnace, 8x12 ft. in area, having a charge hole at one end, into which the liquid slag from the A. S. & R. company's lead furnaces, running 12 to 15 per cent zinc, is poured. Nothing but liquid slag and some skulls are treated in the plant to be described.

At the other end of the furnace is a tap hole which discharges the slag, freed of most of the zinc, into 30-ton pots. The bottom of the furnace is made of water-cooled cast-iron plates, and about 4 in. above these plates are the tuyères. Therefore, the working bottom of the furnace is a layer of chilled slag. The sides of the furnace consist of ordinary water jackets, made rather stronger than in the usual copper or lead blast furnace. This is necessary because the action within the furnace is extremely violent, the turbulence much more resembling that found in a converter than in the peaceable and quiet lead or copper furnace.

The tuyères are 22 in number, 11 being on each side, are 4 in. in diameter, and each tuyère is provided with a ¾-in. coal pipe, through which coal is fed into the blast air immediately before it enters the furnace. The charge hole and charge floor are 15 ft. above the furnace bottom. Above the charge hole is a rectangular canopy of water jackets which confines the gases so that they may be conducted into the dust-collecting system. Then comes a horizontal flue of water jackets, 8x8 ft., and 34 ft. long. This flue is provided with large adjustable openings through which air is admitted to burn the metallic zinc vapors to zinc oxide, also burning the excess coal dust and carbon monoxide.

The rest of the plant is more or less conventional, consisting of the usual system of cooling tubes to bring the temperature of the gases down so that they will not burn the bags. First comes a brick downtake; then two parallel brick flues, each having a cross-section 10 ft. square; then two parallel 8-ft. steel flues each 390 ft. long; then upright U-tubes presenting a cooling surface of about 32,000 sq.ft. The U-tubes, of course, effect the real cooling. They are 32 in number, 38 in. in diameter, and each is 57 ft. high. The system is arranged so that eight tubes are connected in parallel.

Finally comes the baghouse, in which are hung 576 woolen bags, 18 in. in diameter and 30 ft. high. Gases are drawn from the furnace and U-tubes and forced through the bags by a fan 7½ ft. in diameter. The fume drops into hoppers which discharge into gondola cars for shipment to the electrolytic refinery at Great Falls. The entire plant is unique in shape, being 40 ft. wide and 1,000 ft. long.

After crushing the coal in a Raymond mill it is fed into the furnace by four machines, of which one is allotted to each quarter of the furnace, an extra one being provided as a spare. Each machine handles either five or six tuyères. There is a bin in each machine which holds 1½ hours' supply of coal, and at the bottom is a chamber with a screw delivering coal to an outlet in which an injector is placed. This injector is run at a pressure of 90 lb. and gives the current of coal sufficient momentum to carry it through about 20 ft. of pipe connecting the feeder to the furnace. By way of lessening the burden on the injector, a pressure of about 13 lb. is maintained at the top of the coal bin.

On Dec. 3, 1927, the plant was blown in. The first six months were full of troubles which were not met with at Tooele, where no attempt was made to run continuously. The first and worst difficulty was with the bottom. Various kinds of brick were tried, with indifferent success, so the cast-iron water-cooled bottom was finally selected. Some other annoyances had to be overcome by devising special appliances for uninterrupted operation. On the whole, there is now less trouble than one usually encounters in a metallurgical plant.

The slag, to the amount of about 300 tons a day, comes in pots holding five tons, and about five pots make a charge. Sometimes, when the slag supply is not sufficient, skulls are added. The time of blowing a charge runs from 100 to 160 minutes, and the depth of the liquid bath in the furnace is about 3 ft. From 20 to 35 per cent of coal is used, and there must be a slight excess of unburned coal; the amount fed depends on the condition and nature of the slag received. Extraction of zinc ordinarily runs from 85 to 90 per cent. It is easy, by continued blowing, to bring up the extraction to 95 or 98 per cent, but it does not pay. Besides the zinc, any lead left in the slag received is also recovered.

# Arsenicals— An Important Byproduct

By H. C. Gardiner

ARSENIC plays a peculiar and vitally necessary part in its relation to industry and agriculture. Considerable quantities of this material are regularly used in insecticides, in dips, in weed killing, and in wood preservation. Occasional urgent demands occur when material is required to halt the inroads of some pest that threatens crops and the prosperity of communities dependent upon these endangered crops. Were arsenic production a mining operation of itself the sudden urgent demands for large tonnages to take care of emergencies that usually seem remote would be hard to fill and immense damage might occur before quantity production could be achieved from arsenic-producing mines.

Irrespective of immediate market demand, arsenic is produced as a by-product of copper-smelting operations, and large tonnages accumulate. A few years ago grasshoppers threatened the agricultural wealth of a province in the Canadian Northwest, in a section of the country where such a pest was practically unknown. Within a few days ten 50-ton carloads of white arsenic were dispatched from Anaconda as a munition of war against this menace. When the boll weevil becomes serious in the South and the cotton crop is threatened, thousands of tons of accumulated byproduct arsenic become immediately available for manufacture into a potent remedy for this pest.

Arsenicals are used to poison insects, being either sprayed or dusted on fruits, vegetables, or foliage on which insects feed. White arsenic, calcium arsenate, lead arsenate, and sodium arsenate are extensively used in preparing poison baits for insect pests. Dips for cattle and sheep are also prepared from arsenic. More than 21,000,000 lb. of white arsenic was used in 1928 for these purposes.

Not only is the cotton crop of the South protected from the boll weevil, but arsenic protects alfalfa from the alfalfa weevil, and to the orchards of California, Florida, New York, and other important fruit-producing regions, protection is afforded by arsenicals of various kinds.

During 1927, 27,000,000 lb. of lead arsenate was used in the United States,

chiefly in poison sprays and dusts in destroying insects which injure fruits and vegetables, such as the codling moth, plum curculio, cabbage worm, potato bug, and nearly a dozen other similar pests. The same quantity of calcium arsenate was used against the cotton boll weevil, the alfalfa weevil, and other insects affecting fiber and forage crops.

Effective to the highest degree, arsenic compounds protect wood structures against fungus and insect destroyers. Widely used in Australia and South Africa for many years, arsenic compounds are effectively preserving mine timbers, telephone and telegraph poles, fence posts, and railroad ties against decay. Several million pounds are now used annually for these purposes in the United States. Arsenic compounds also play an important part as weed destroyers along railroad rights-of-way.

The Anaconda company produces annually approximately 50 per cent of the total quantity of white arsenic produced in the United States, which has nation-wide distribution in the agricultural industry.

# French Process Modernized by Anaconda in Production of Zinc Oxide at East Chicago

By F. O. CASE

*Manager, Anaconda Zinc Oxide Department*

MODERN electrochemistry plus an old metallurgical process—thus, in brief, is outlined the production of Anaconda zinc oxide. The raw material, Anaconda electrolytic zinc, is shipped to East Chicago, Ind., and Akron, Ohio, at which the plants are situated for converting it into zinc oxide by the French process.

Zinc oxide is a very white pigment, and, as referred to in this article, is a commercially pure oxide of zinc. Commercially, zinc oxide is manufactured by two processes. The older and indirect process is called the French process; the other and newer is the American process. In the former, zinc metal is distilled in suitable furnaces and the vapor oxidized. In the American process, zinc ore is reduced by the carbon in hard coal or coke and the resulting zinc vapor is immediately volatilized and oxidized to zinc oxide. Each process has its disadvantages; the French process is usually limited on account of cost, especially where low-grade oxides are produced; the American gives a low recovery of zinc and also involves contamination of the finished oxides.

The production of Anaconda zinc oxide by the French process was started in December, 1921, with a small pilot unit at East Chicago. The first commercial unit was erected there early in 1922 and started operation in June. Also, a duplicate was erected the same year at Akron to serve the rubber industries. Akron started operation in January, 1923, and during the summer of that year the capacity of each plant was more than doubled. Except for minor improvements and repairs, all units have been operating continually since completion.

Before Anaconda's work it was assumed by many that zinc oxide of this character was not suitable for paints nor use in rubber compounds, but the results obtained by our customers during the last seven years have amply demonstrated the fallacy of this belief. When Anaconda electrolytic zinc is used as a raw material, the resulting oxide is not only chemically purer but physically better adapted for use in paint and rubber than American process zinc oxide. Paints made from French process zinc oxide are not only whiter but wear better than those made from American process oxide. Rubber compounds are superior because the finer grain size of the French process oxide gives greater resistance to abrasion.

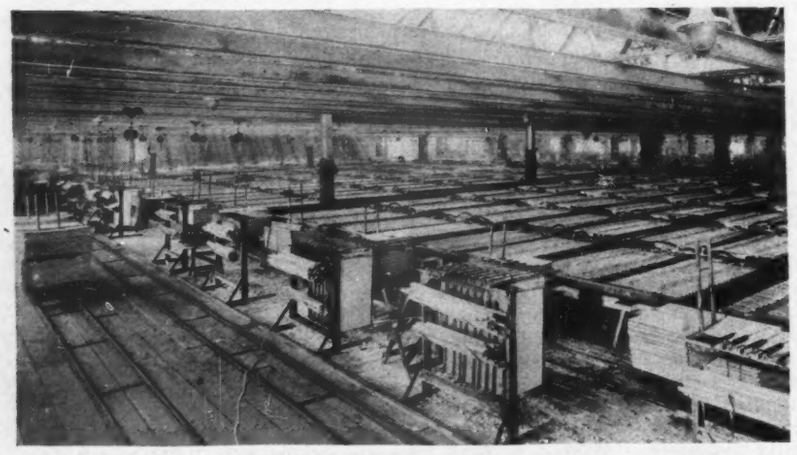
The French process as carried out at the Anaconda plants differs from the older practice not only in the construction of the furnaces and other equipment, but also because all parts of the manufacturing cycle are variable and are under control. Fuel oil furnishes the heat for the furnaces, and the temperature is always uniform, depending upon the grade of material being produced. The rate of oxidation of the zinc to zinc oxide is controlled by regulating both the amount of zinc vapor produced and the volume and temperature of air used for oxidation. This air is moved and controlled by large Buffalo Forge fans connected to variable speed Reeves controls. The temperature and humidity of this air are also kept uniform.

After oxidation, the mixture of zinc oxide and air is cooled, passed through settling and classifying chambers, and then the zinc oxide is filtered out by passing the oxide-laden gases through muslin or woolen bags. A rough classification of zinc oxide into various particle sizes is made by the settling chambers and baghouses.

By selection of raw material and suitable control in the manufacturing cycle, Anaconda makes the following grades of zinc oxide: Pharmaceutical (U.S.P.), White Seal, Green Seal, Red Seal, Selected and Special Lead-Free, Lead-Free Zinc Oxide with a very fine grain size, and ordinary Lead-Free Zinc Oxide. The Pharmaceutical grade is used in cosmetic and medicinal preparations. Red, Green, and White Seal grades are used in enamels, lacquers, printing ink, and other products demanding various properties found only in a high-grade zinc oxide pigment. The Lead-Free grades are used in paints and rubber compounds. Extremely fine-grained zinc oxide is used in rubber compounds where great strength and resistance to abrasion are needed, such as heavy-duty high-grade tires.

The production and sale during the last seven years of large tonnages of Anaconda zinc oxides have not only overcome the prejudice against French process zinc oxides, but have clearly shown their superiority. This is the logical and obvious result of using a superpure raw material — Anaconda electrolytic zinc—and a controlled manufacturing process that prevents contamination of the finished product.

**Interior of electrolyzing division at Great Falls, where the pure metal for the East Chicago plant is precipitated**



# Mining the World's Largest Copper Ore Deposit at Chuquicamata, Chile

By H. C. BELLINGER
*Vice-President in charge of operations, Chile Copper Company*

Burr Wheeler

H. C. Bellinger

INASMUCH as the operations at Chuquicamata have been described in detail in different articles in the technical press, I shall give only a short outline of our operations and mention later developments or changes that have taken place since the above-mentioned articles were published.

It may be of interest, to those who are not acquainted with the early history of Chuquicamata, to know that occasionally during our mining operations, primitive tools have been uncovered, such as stone hammers and wooden shovels, and even the mummified remains of early miners, one of which is on exhibit at the American Museum of Natural History in New York City. About 25 miles from Chuquicamata are found the ruins of the fortified village of Pucaro, which was destroyed by the Spaniards at the time of their first invasion, and in the graves pertaining to this village have been found bowls and other copper utensils, and bead necklaces made of brochantite and atacamite, typical Chuquicamata minerals; all of which indicate that the Chuquicamata ore deposit was worked several hundred years ago, probably some time between the Inca conquest of Chile, which occurred about 1443, and the first Spanish Invasion ninety years afterward.

When the present Chilean provinces of Antofagasta and Tarapaca were ceded to Chile in 1879, at the close of the war between that country and the allied nations of Peru and Bolivia, mining activities at Chuquicamata increased, operations being carried on by Chilean and British companies, which mined the surface ores and also the higher-grade ores in the veins. Of course these operations were carried on in a relatively small way. In 1910, A. C. Burrage became interested in the deposit, and he, in turn, interested the firm of M. Guggenheim Sons, which firm, in 1912, after investigation, formed the Chile Exploration Company, and prospect drilling of the deposit was started in that year.

Construction of the first unit of the present plant was begun in April, 1913, and started operating in May, 1915. Its capacity was 6,500 tons of ore per day, which was gradually expanded so that by 1923 the capacity was 22,000 tons of ore per day, with an estimated productive capacity of 225,000,000 lb. of copper per year. On March 1, 1923, control of the Chile Copper Company and its operating subsidiary, the Chile Exploration Company, passed to the Anaconda Copper Mining Company, since which time further additions to plant and equipment have brought the capacity up to 400,000,000 lb. of copper per year.

The character and size of the orebody are of such a nature that by comparable additions to the plant, production could, say within two years, be brought up to 500,000,000 lb. of copper per annum from the oxide ores alone; and with the coming in of the sulphide ores and the construction of a plant for the treatment of these ores, production can be increased to 600,000,000 or 700,000,000 lb. per annum without difficulty.

The officers of Chile Copper Company, together with the operating subsidiary, Chile Exploration Company, are John D. Ryan, Chairman of the Board; Cornelius F. Kelley, President, B. B. Thayer and H. C. Bellinger, Vice-presidents, and C. W. Welch, Secretary and Treasurer. Anaconda's metallurgical, mechanical, and electrical departments, headed by Frederick Laist, are consultants to the company. Burr Wheeler is General Manager in charge of all operations in Chile, and E. J. Craig, Resident Director of the Chile Exploration Company, is General Representative in Santiago, Chile, of the Anaconda Copper Mining Company and its subsidiaries operating in Chile.

Ore reserves as calculated from the original boreholes, both positive and probable, amounted to 700,000,000 tons of oxide, mixed, and sulphide ore of an average grade of 2.12 per cent copper. During the last few years we have put down a further number of exploratory holes which have developed additional ore. This program of exploration is still being carried out, and no attempt will be made to recalculate the ore reserves on the basis of the new discoveries, until the drilling campaign has been completed. It may be stated, however, in a general way, that the amount of new ore developed equals or nearly equals the amount of ore so far removed.

In the early years of our operations the "cut-off" between ore and waste was fixed at 0.7 per cent copper, and the ore-reserve estimates were based on that "cut-off." Improvements in practice, both mining and metallurgical, resulting in lower costs than were expected, have allowed a very material reduction in this "cut-off" grade. The lowest grade of material that can be treated without loss is about 0.23 per cent copper, based on a 13c. copper sales price, and provided such material must be removed in mining operations. Naturally, the treatment of any considerable tonnage of such a low grade would be economically unsound. However, in May, 1926, it was determined, as applying to material that occurs within the eventual slopes of the open pit and must therefore be removed in mining operations, that the most economical "cut-off" between ore and waste was 0.45 per cent copper, with a 13c. copper sales price. However, this minimum grade may vary with other conditions, such as the relation between tonnages of

**A part of the Chuquicamata plant, taken July 14, 1928, just after a snowstorm, an extremely rare occurrence there**

higher-grade and lower-grade ores as contemplated in the mining program over any stated period. Such a reduction in the "cut-off" grade will increase the estimates of ore reserves, when they may be recalculated.

The mine at present consists of thirteen benches, varying in height from 37 to 64 ft., the average being about 50 ft.; and the total footage of faces exposed amounts to 60,000 ft., or 11.4 miles.

At the start of our mining operations, churn drilling was tried, but it was quickly found that this method of blast development was uneconomical, and, in fact, impractical, on account of the unknown extent and ramifications of the early underground workings in the upper portion of the orebody. In consequence, the tunnel-blasting method was decided upon as the only possible one that could be satisfactorily used at that time. After the superficial zone of the antique workings had been mined, churn-drill hole blasting proved a more satisfactory method of breaking the ore, and blasting is now accomplished almost entirely by the use of churn-drill holes, tunnels being employed only where they have previously existed.

During the early years of our operations, all churn drills, shovels, and locomotives were steam operated, but gradually, with the development of electrically operated equipment, we discarded steam machines, and the substitution has in every instance resulted in a most satisfactory saving in costs. The first electric churn drill was introduced at the property in 1919, and today we have in operation 64 electrically actuated churn-drill rigs. The first electric shovel was placed in operation at the property in 1918. This shovel proved its great economy over the steam shovels, so additional electric shovels were purchased. By 1922 all of the steam shovels had been replaced. The first electric shovels that we purchased were of alternating-current type, but the practice during the last seven years has been in favor of direct-current machines under Ward Leonard control. Our shovel equipment now consists of nineteen four-yard capacity, three eight-yard capacity, and three three-quarter-yard Diesel shovels. The eight-yard full revolving shovels are used for bench development and the four-yard shovels for general loading operations. The small Diesel shovels have been found highly satisfactory for cleaning up on the benches, preparing roadbeds for tracks, and other miscellaneous uses. All shovels are equipped with caterpillars.

Electrification of our transportation system was started in 1926, and at present electric motive power is used for transporting the major portion of our ore. The complete electrification program, which covers not only the mine but the plant and the main lines as well, we hope to have completed within the next year.

Combination storage-battery and third-rail locomotives serve the mine and the plant, power being derived from storage batteries while the locomotives are on the benches and from the third rail while on the main lines. The storage-battery feature of the locomotives was introduced for two reasons: first, safety to our workmen, and, second, to provide greater flexibility of operation while on the benches, owing to the rapidly changing location of the bench faces and the consequent moving of tracks. The locomotives used on the tailing-disposal system are designed for third-rail and trolley service, the trolley system being used only on the outer rim of the progressively increasing face of the dump where the use of third rail would be impractical. A total of fourteen electrically operated 76-ton locomotives are in use at present.

A comparatively new feature of our blasting operations is the use of liquid oxygen in place of high explosives. In 1926 one 75-liter-per-hour unit was placed in commission, and the result of the operation of this plant indicated that a material saving could be achieved in our blasting cost. Two additional units have been added, each with a rated capacity of 80 liters per hour, but in actual operation we have found that these two units are capable of delivering regularly an average of 90 liters per hour of 99.5 per cent pure liquid oxygen. It is our intention soon to add sufficient additional units to replace practically all of our high-explosive requirements with the exception of the powder used for blockholing. We continue to use black powder in the upper and less hard portion of the orebody, where this procedure is the more economical.

Other cost-reducing features are employed, such as mechanical spreader plows for use on dumps, and mechanical track shifters, both of which have made a material reduction in the number of men required for this work, and a consequent saving in cost. In 1927, the plant staff designed a car-pulling device which is attached to the shovel and with which all four-yard shovels are now equipped. This device has materially reduced locomotive service to the shovels because of the elimination of the dead time previously resulting from the locomotives standing idle while ore and waste trains were being loaded. We have a central control at the mine for locomotive service, with which all operating shovels are in communication, and on instructions from the shovels, locomotives are dispatched in sufficient time to avoid operating and transportation delays. A machine shop on wheels has also been provided, containing sufficient tools and equipment to enable the making of field repairs to drills, shovels, and other equipment.

William H. Goodrich, in an article entitled "Mining Operations at Chuquicamata, Chile, South America," in the December, 1926, issue of the *Mining Congress Journal*, explains in detail our method of churn-drilling with reference to spacing and depth; and discusses the handling of explosives, control of grade of ore, and transportation. H. C. Schultz and F. K. Middleton Hunter, in Technical Publication No. 61 of the A. I. M. E., 1928, have described in detail the operation of the first unit of our LOX installation and the use of this explosive in actual operation. Both of these articles fairly represent our practice today, with the exception of the substitution of electrically operated locomotives and shovels for such steam equipment as is mentioned.

The crushing department is divided into two sections: the West plant and the East plant, the West plant being the original installation and the East plant an addition placed in operation the latter part of 1927. Both plants are complete units in themselves—that is, from coarse to fine crushing.

The West primary plant is served by one movable car dumper, and consists of two 84x60-in. jaw crushers. In the West secondary plant are four No. 10 McCully gyratory crushers, eight Symons 48-in. coarse reduction disk crushers divided into four sections of two crushers each, and twenty-four Symons 48-in. fine reduction disk crushers, twelve running parallel with the north side of the building and twelve with the south side. Since the East

primary plant, which has sufficient capacity to cover our present requirements, was placed in operation, the West primary has stood as a reserve unit.

East primary plant is served by two revolving car dumpers of 105-ton capacity, the ore cars being delivered to the dumpers from the storage yard by pusher locomotives. This plant is equipped with two 60-in. McCully double-discharge gyratory crushers. The East secondary plant is divided into three sections, each section consisting of one No. 10 McCully gyratory crusher, two 48-in. Symons vertical coarse crushers, and four 48-in. Symons vertical fine disk crushers.

The ore is crushed to pass 0.371-in. square mesh, the oversize amounting to about 10 per cent. All ore is transported between crushers and from crushers to leaching vats by conveyor belts ranging in width from 36 to 60 in., the greatest length being about 1,100 ft. A central dust-collecting house serves both the West and the East crushing plants.

Two sampling towers are provided, one serving the West and one the East crushing plant, in which samples of the feed to the leaching tanks are taken at regular intervals. Tailing samples are also handled in these towers.

The leaching division at present consists of thirteen tanks, each 150 ft. long and 110 ft. wide. Six of these tanks have a depth of 16½ ft., and the remaining seven a depth of 18½ ft. Capacity of the smaller tanks is 10,243 metric tons each and of the larger tanks approximately 11,000 tons each. Four days is the average soaking cycle for each charge. The solutions are handled through wood-stave pipes, all of a diameter of 15 in., with the exception of two lines, which are 24 in. in diameter. Leaching vats 1 to 6 are equipped with one filling bridge and two discharging bridges, and vats 7 to 13 are similarly equipped.

Each of the eighteen solution sumps is 150 ft. long, 12 ft. deep, and of varying widths, ranging from 35 to 130 ft. Capacity of all solution sumps is approximately 57,000 cubic meters. All solution sumps and leaching tanks are of reinforced concrete and lined with a special mastic mixture. To handle the solutions, where the flow is not by gravity, three pump houses are provided, in which are installed a total of eighteen 15-in., four 9-in., two 6-in., one 4-in. and two 3-in. electrically driven pumps. Tailing is loaded into 12-yd. and 20-yd. cars and hauled to the dump by electric locomotives.

The building housing the electrolytic department is approximately 160 ft. wide by 1,188 ft. long. Each of the 1,098 tanks, of which 64 are in the starting-sheet division, is 19 ft. 2 in. long by 3 ft. 11 in. wide by 4 ft. 10 in. in depth (inside dimensions). Forty of the starting-sheet tanks are used for producing starting sheets from copper anodes, and the other twenty-four for producing starting sheets from commercial solution. Tanks are arranged in sections of sixteen each, with a few exceptions where there are ten or seventeen tanks to a section. The sections are divided into circuits containing from 96 to 112 cells. Tanks of all sections are arranged in cascade, the solution flowing through each tank of a section by gravity. As with the leaching department, wood-stave pipe is used exclusively throughout the electrolytic department.

The original solution as received from the treatment of the ore—called the strong solution—is first dechloridized and then delivered to the head tank, located just outside of and at a higher elevation than the electrolytic tank house. The solution reaching this head tank carries from 40 to 60 grams per liter in copper, and it is here diluted with spent electrolyte to a point where the copper content averages about 24 grams per liter, after which it is passed through the tank house.

Spacing of anodes and cathodes in the cells is 4 in. and each cell contains 56 anodes and 55 cathodes. The anodes weigh approximately 250 lb. and the cathodes, on pulling, weigh approximately 150 lb. C. W. Eichrodt, in a paper entitled "The Electrolytic Tank House, Chile Exploration Company, Chuquicamata, Chile," presented at the meeting of the American Electrochemical Society held in April, 1924, gives a full and complete description of our tank-house procedure, as well as of the manufacture of our anodes and starting sheets.

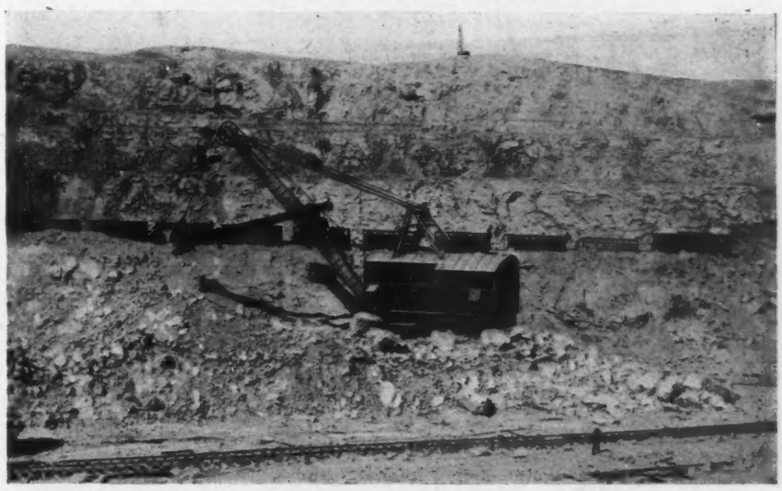
The market furnaces, of which there are three, are all of approximately the same size, the hearth length being 49 ft. 5 in., width 13 ft. 10 in., and depth 7 ft. 6¾ in. Each furnace has three charging doors and is equipped with a Clark casting wheel. Furnace construction consists of a bottom of concrete, with air ducts for cooling; and a course of 12-in. silica brick overlaid with a course of 22-in. silica brick. Side walls are of 9-in. silica brick outside, and 13½ in. of magnesite brick inside, with 12-in. silica brick roofs.

The power plant is at the port of Tocopilla, Chile, a distance of about 140 kilometers from Chuquicamata. Boiler installation consists of 24 Babcock and Wilcox oil-fired boilers, sixteen of which are rated at 595 h.p., two at 1,351 h.p. and six at 1,403 h.p. The total generating capacity of the station is now 90,000 kw., with five 10,000-kw. and two 20,000-kw. units. In addition, there is one 3,000-kw. turbo-generator for local service. Output from the main generators is 3-phase, 50 cycle, 5,000 volts, which is transmitted to the plant at Chuquicamata at 110,000 volts. The present transmission system consists of three lines, by 110,000 volt, 3-phase, 50-cycle. All fresh water used for boiler feed and drinking purposes is evaporated from sea water.

Fuel-oil storage capacity amounts to 300,000 bbl. The company owns and operates, through its steamship subsidiary, Chile Steamship Company, the freighter "Chilcop" and three tankers, the "Chiloil," "Chilsco," and "Chilbar," and in addition has under charter a number of other tankers. The fleet of tankers serves the power plant at Tocopilla and also delivers oil at Antofagasta for use at Chuquicamata and at the port of Barquito, Chile, for the operations of the Andes Copper Mining Company. After the oil for the mine and mill is delivered at Antofagasta, it is transported to Chuquicamata by rail in tank cars.

The company houses all of its employees at Chuquicamata and Tocopilla. At present we employ about 6,000 men at Chuquicamata, and with their families this requires the housing of about 18,000 people. At Tocopilla, the company employs 450 men and furnishes adequate housing accommodations for most of these men and their families. A main hospital, up to date in every way in its equipment, is provided by the company at Chuquicamata for the benefit of all its employees. This hospital contains four general wards and one women's ward as well as private accommodations, with a total of 125 beds. In the workmen's village is a 25-bed maternity hospital, which has proved a boon to the families of our

**An 8-yard electric shovel loading ore into cars on bench above**



workmen. The staff of physicians at Chuquicamata consists of a surgeon in charge, an assistant surgeon, and four physicians. Five foreign nurses are on the staff and about twenty-five Chilean nurses. The company also maintains medical facilities in charge of a physician at Tocopilla.

In addition, social clubs, athletic fields, and other forms of recreation and amusement have been provided by the company. Schools for the children of the English-speaking employees are maintained at both Chuquicamata and Tocopilla, also at the company's expense. At Chuquicamata, schools are provided by the company for the children of the Chilean employees. During 1928, the average attendance at these schools was 1,130 pupils. The salaries of the staff of teachers at the Chilean schools are borne by both the Chilean government and the company. We also maintain at Chuquicamata a trade school, at which instruction is given to employees in the various mechanical arts, free of cost.

All copper and practically all equipment and supplies for the plant at Chuquicamata are shipped and received through the port of Antofogasta, Chile, 261 kilometers from Chuquicamata. There an office is maintained to handle these shipments and also make a small amount of local purchases. At Santiago, the capital of Chile, the company maintains an office in conjunction with the Andes Copper Mining Company and other members of the Anaconda group, through which are handled legal questions, taxation, and other matters requiring consultation with officials of the central government of Chile. Arrangements for providing funds for payrolls and local purchases are also a function of this office. At Valparaiso these same companies have an office, under the direction of the Santiago office, through which are purchased food supplies and such plant supplies and equipment as are obtainable from domestic sources in Chile.

# Manufacturing the Raw Material

## *American Brass Company Has Seven Strategically Located Plants*

By JOHN A. COE

*President, American Brass Company*

THE AMERICAN BRASS Company, through the units which have entered into its organization from time to time, dates from the very beginning of the brass and copper industry in this country. The oldest and one of the most important of these units began operations in 1812, in Waterbury, Conn., and was probably the first concern to furnish sheet brass to the market, in addition to supplying its own requirements in the manufacture of metal buttons. As the business increased, other concerns entered the field and located either in Waterbury or in near-by towns in the Naugatuck Valley, where adequate water power was available. During the forty years subsequent to 1812, four of these Naugatuck Valley concerns were organized; in 1834 the Wolcottville Brass Company, in 1844 the Ansonia Brass & Copper Company, in 1846 the Waterbury Brass Company, and in 1853 Holmes, Booth & Hayden, all of which grew to considerable proportions, and were later merged into The American Brass Company.

It will be seen, therefore, that the Brass company is built upon a basis of practical experience in the manufacture of brass and copper materials that is unequaled by any other aggregation in that line of business in the United States. The concerns named are the oldest members of the corporation, but several newer units have been added since the original organization of the company, so that all periods of development of the brass and copper industry for nearly a century and a quarter are represented in The American Brass Company fabric as it stands today. These large, completely equipped plants are located at Ansonia, Torrington, and Waterbury, Conn.; Buffalo, N. Y.; Detroit, Mich.; Kenosha, Wis.; and New Toronto, Ont., Canada; locations that have been planned to serve industry in various sections of the continent with the least expense for and delay in transportation. About 16,500 people comprise the company's payroll at these various plants.

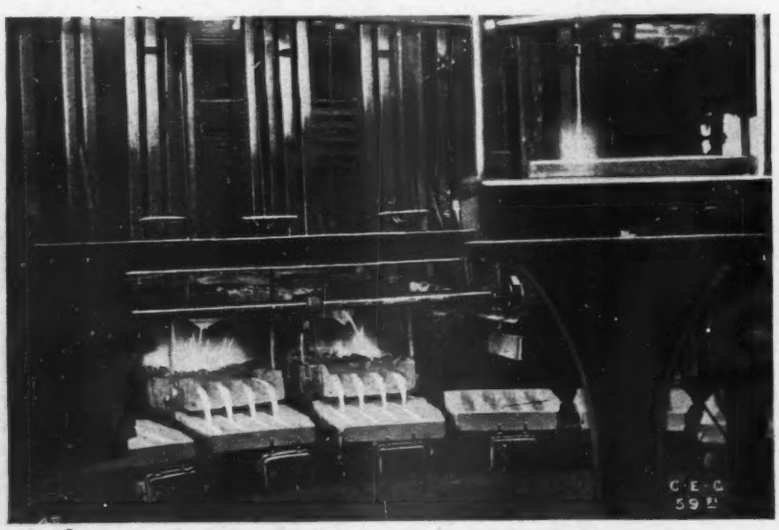
In the early operations of brass and copper mills the uses for their products were limited; brass buttons, brass kettles, clocks, pins, hooks and eyes, and later, after the discovery of petroleum, brass lamp burners. Builders' hardware, in the early days made to a large extent from iron, gradually came to be manufactured chiefly from brass and consumed considerable amounts of metal. However, in recent years numberless uses have developed of which the early copper manufacturers had no conception, and the magnitude of the business has become such as to make the tonnage of the early years insignificant.

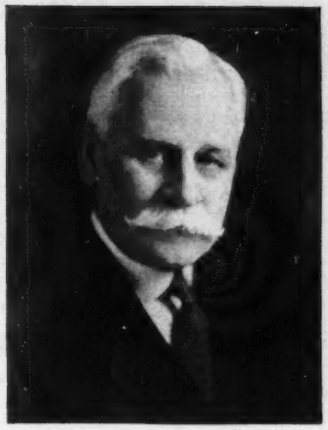
In increasing copper consumption, the development of electricity has probably been the largest single factor, but the amazing increases in the use of the automobile, refrigerating devices, and washing machines, to mention only a few outlets, account for a large and ever-increasing consumption of copper products. Among the most noticeable items of brass consumption in recent years may be mentioned brass pipe for plumbing, and copper for roofing and rain conductor purposes, as well as all manner of automobile accessories, electrical devices in great number, and an endless variety of screw-machine products, consuming millions of pounds of brass rods yearly.

The product of the company covers the entire field of copper and copper alloys, particularly the various brasses and bronzes, and varies widely both as to composition and as to the form in which it is manufactured. Various forms of pure copper as obtained from the refineries comprise only a portion of its output. In its own melting shops copper is combined in the electric furnace with such other constituents as zinc, nickel, tin, and lead, in varying proportions to form a wide range of alloys in constant commercial use, and for special purposes alloys containing aluminum, silicon, manganese, and cadmium are regularly produced. Altogether, the active list of alloys manufactured in one or more forms of wrought material numbers upward of two hundred.

An entirely new series of alloys has

**Casting wirebars from 300-ton refining furnace**





John A. Coe

recently been made by alloying copper with silicon. Silicon strengthens copper, its effect being similar to that of tin, but more pronounced, and the copper-silicon alloys can be used to replace bronzes both in wrought metal and in castings. The company's trademarked alloy Everdur, which is copper 96 per cent, silicon 3 per cent, and manganese 1 per cent, has a high resistance to corrosion and when cold-rolled or drawn has a tensile strength of 75,000 to 100,000 lb. per square inch. Another new silicon-copper alloy containing copper 95 per cent, nickel 4 per cent, and silicon 1 per cent is readily hot-rolled or forged, and is ductile and easily wrought by cold-working when quenched from temperatures above 750 deg. C. Such treatment places the nickel and silicon in solid solution in the copper, and in this condition it has a tensile strength of approximately 50,000 lb. per square inch. By heat treating at 450 deg. C. for two or three hours, the nickel silicide is precipitated out of solution and the alloy has a tensile strength of about 100,000 lb. per square inch with an elongation of 16 per cent. A copper-cadmium alloy for high strength conductors has more recently been developed. This is hot-rolled and cold-drawn and makes wires with a tensile strength up to 80,000 lb. per square inch which still have a conductivity of 89 per cent.

The methods of manufacturing are many and varied. Pure copper and alloys in sheet, plate, or strip form are produced from slabs by hot- or cold-rolling or a combination of the two methods according to their nature, the rolling reduction itself being supplemented at various stages by the softening effect of furnace annealing. The rolled product is oftentimes subjected to further surfacing treatment, such as polishing, tinning, or nickel-plating, before it leaves the mill. The range of size in this material runs from 10,000-lb. condenser head plates of 1½ in. thickness to shim brass stock as thin as 0.002 in.

Copper and alloy tubing is made by any one of four primary processes: cored casting in the form of pipe, hot-piercing a solid billet to form a tube, cupping a disk of sheet metal, and tubular-forming strip metal, then brazing the lengthwise seam. By whatever process made, the primary product is subjected to cold-drawing and annealing operations, which refine the grain, polish the surface, and produce the desired dimensions in wall thickness and diameter. The usual range of diameters is from a maximum of 12 in. to a minimum of $\frac{1}{4}$ in.

Rods and bars of various shapes, whether of copper or its alloys, are produced by hot- or cold-rolling on rolls containing shaping grooves or by forcing the heated metal through a die under powerful hydraulic pressure, known as the extrusion process. In either operation, the material usually receives its final sizing and correct temper by being cold-drawn to exact dimensions. The usual sizes cover the range from a diameter of 6 in. or its equivalent cross-sectional area to a diameter of $\frac{3}{32}$ in.

Wire is produced by the same primary processes as rods and bars; that is, by hot- or cold-grooved rolling or extrusion, and differs mainly from rods in that it is afterward subjected to more extensive die-working in long coils, requiring the use of special machines designed for multiple drawing operations at high speed, interspersed with accurately controlled annealing operations. The usual sizes run from trolley wire to diameters as small as 0.002 in.

Such semi-finished fabricated parts as are made by the company in its own or allied plants are mainly produced by hot-forging or cold-press working, throughout an extensive range of materials from valve bodies to collar buttons.

Owing to the complexity of its manufacturing problems, involving the melting, heat-treating, mechanical shaping, and special surfacing of its products, the company has been foremost in devising and adopting improved equipment and methods to meet changing conditions in the production of non-ferrous materials. It introduced the hydraulic extrusion process and Mannesmann billet piercing in this country and pioneered in the utilization of electricity for melting as applied in the induction and high frequency melting furnaces and their application to the heat-treatment of metals. It has always maintained an extensive engineering staff for the study and improvement of its equipment, which has been responsible for much of the mechanical progress made in the industry; and, in the same way, the company was the first in the industry to establish a metallurgical research department, and to engage actively in systematic development of improved properties in its materials.

The foregoing gives in condensed form a general statement of the formation, progress, and present activities of The American Brass Company, as well as some idea of the growth and history of the copper and brass industry in the United States since the early part of the nineteenth century. Although the mining and refining of copper and the fabrication of the metal have been carried on in the past as separate industries, a tendency toward consolidation has developed within the last few years, the first conspicuous instance of this kind being the absorption of The American Brass Company by the Anaconda Copper Mining Company. Other developments of the same nature have followed and still others may appear in the future. It is believed that such consolidations will be for the ultimate improvement of the non-ferrous metal industry by reason of the production of a better product, owing to complete continuous supervision from "mine to consumer." Certainly the progress of the Anaconda Copper Mining Company thus far since the consolidation seems to support this view.

**Sheet copper department of the Buffalo mill**

# Leaching, Flotation, Smelting Included in Andes' Operations at Potrerillos, Chile

By WILLIAM WRAITH
*Vice-President, Andes Copper Mining Company, New York*

THE Andes Copper Mining Company's properties are situated in the central part of the Republic of Chile. The company's port, Barquito, lies about two miles south of the port of Chanaral, and is equipped with the necessary facilities for receiving cargoes from and delivering copper to ocean-going steamers. Here also is located the company's main power plant, from which electric power is transmitted 78 miles east to Potrerillos, the site of the reduction works. Potrerillos lies at an elevation of 9,500 ft., and is connected with Barquito by a 1-meter gage railroad 94 miles long, of which approximately 58 miles is owned by the Andes company and the remainder by the Chilean State Railways, with which the company has an agreement for the operation of its trains.

The mine lies to the southeast of Potrerillos at an average surface elevation of 10,500 ft. A 1-meter gage railroad, 6.2 miles long, connects the underground mine-storage bins with the reduction works at Potrerillos.

*Ore Deposit*—The Potrerillos ore deposit consists of the copper-enriched portion of a plug of quartz-diorite porphyry which has intruded overlying sedimentary rocks. The outcrop of this intrusion is approximately 1¼ miles long, with a maximum width of about ⅝ mile, and of this area the ore deposit, as now developed, measures approximately 4,900 ft. in length by 2,500 ft. in maximum width. Average thickness of the ore as shown by churn drilling is 274 ft., and the deposit has been estimated to yield 137,400,000 tons of ore having an average copper content of 1.51 per cent. Of this tonnage approximately 35 per cent is oxide ore.

*Mining*—The horizon of the orebody is such that no hoisting is required. Underground storage bins, at the mine terminus of the mine-to-mill railroad, are situated directly below the central part of the orebody, and inclined ore-passes connect the two intermediate haulage levels of the mine with these storage bins.

Under the method of mining employed, the ore is undercut and caved at suitable horizons, drawn through a series of branching raises, and delivered by gravity through raises to chutes on the intermediate haulage levels. On these levels the ore is loaded into trains of sixteen or twenty five-ton cars, which are hauled by ten-ton trolley locomotives to pockets at the upper ends of the ore-passes, into which the ore is dumped and then passed by gravity to the mine storage bins. Details of the mining practice were recently published by the American Institute of Mining and Metallurgical Engineers, as Publication No. 155, by I. L. Greninger.

*Mine-to-Mill Railway*—The mine storage bins are located near the end of a haulage adit 2.17 miles long. Ore is loaded from the bins into cars having a nominal capacity of 40 tons, and trains of 25 cars are hauled by two 45-ton electric locomotives, in tandem, to the crushing plant storage bins. Current is transmitted to the locomotives by a third rail.

*Sulphide Treatment Plant*—This plant has a capacity of 15,000 tons of ore daily and comprises the customary crushing, concentrating, calcining, smelting, and converting units. At the primary crushing plant the ore is reduced to 3½ in. by No. 9 gyratory crushers, and is delivered by conveyor to the secondary crushing plant, where it is reduced to 1 in. by 48-in. disk crushers. Then the crushed ore is delivered by conveyor to the concentrator, where it is ground in 9-ft. diameter ball mills and subjected to flotation concentration. The concentrate is delivered to 100-ft. diameter thickening tanks, and the thickened product is filtered by 12x14-ft. diameter vacuum filters. The product from these, with proper fluxes, is delivered by conveyor to eight-hearth, 22-ft. diameter roasting furnaces, from which part of the product, in the form of dried concentrate, is trammed directly to the reverberatory furnaces. The rest of the dried concentrate is delivered to a similar set of roasting furnaces for the production of sulphur dioxide gas for acid manufacture.

Calcine from the latter roasters, with the above-mentioned dried concentrate, is charged to 21x115-ft. reverberatory furnaces equipped with the usual waste-heat boilers. The matte from the reverberatory furnaces is transferred by a 60-ton crane to 26x12-ft. Peirce-Smith converters. Blister copper is transferred by crane to 18x8-ft. diameter tilting furnaces, from which the copper is poured in cakes for shipment to the electrolytic refinery of the Raritan Copper Works, at Perth Amboy, N. J.

*Oxide Treatment Plant*—This has a daily capacity of 7,500 tons of ore, the process comprising leaching the crushed ore with sulphuric acid, purification of the resultant copper-enriched solution, electrolytic precipitation of the copper therefrom, and melting of cathodes and casting into commercial shapes.

The ore is reduced to 3½-in. size at the primary crushing plant by No. 9 gyratory crushers, then being delivered by conveyor to the secondary crushing plant, where it is reduced in two stages to ¼ in. by 48-in. disk crushers. Preceding the first stage the ore is screened for removal of fines. Undersize is subjected to hydraulic classification, the

**Pouring the first copper from the converter at Potrerillos, Jan. 14, 1927**

sand being returned to the screened ore going to the leaching tanks. The classifier overflow will be treated by special leaching methods, not yet installed.

After screening, the oversize is delivered by conveyors to 7,500-ton leaching tanks and subjected to a six-day, counter-current acid treatment, followed by the usual counter-current washing. The strong copper solution obtained is pumped to the purification plant, where it is mechanically agitated with ground limerock. Following this procedure the contents of the agitation tanks are transferred to other tanks heated by steam coils and then to a Moore filter operation, where the precipitated impurities deleterious to electrolytic operation are removed. The filtrate, which has been neutralized in the course of purification, is restored to a proper acidity and delivered to the dechloridizing plant, where the chlorine in the solution is removed by the use of cement copper as in the process developed by the Chile Exploration Company at Chuquicamata, Chile. The dechloridized solution is further acidified and sent to the electrolytic plant for deposition of cathode copper, which is melted in a 12x29-ft. furnace and cast into commercial shapes.

*Sulphuric Acid*—Sulphur dioxide gas produced from roasting dried flotation concentrate is passed through plate-type Cottrell treaters, and then through Glover towers, followed by a series of brick-packed cells and the usual Gay-Lussac towers. The capacity of the plant is 275 tons of 60-deg. acid per day.

*Power*—Power is obtained from three sources: First, a seacoast plant at Barquito having an installed turbine capacity of 30,000 kw., the current from which is transmitted at 88,000 volts to the substation at Potrerillos. Second, a hydroelectric plant of 2,000-kw. capacity installed in the La Ola pipe line to absorb surplus hydraulic head. Third, a 3,750-kw. turbo-generator operated by steam developed by the waste-heat boilers attached to the reverberatory furnaces.

*Water Supply*—The reduction works' water supply is obtained from the Rio La Ola, a stream that has its source in a series of swamps, or "vegas." Water is taken from a small storage reservoir (elevation 11,650 ft.) into a 36-in. diameter, riveted steel pipe line, 32 miles long, and delivered to the reduction works' storage reservoir at an elevation of 9,720 ft. The pipe line is characterized by the utilization of a number of inverted siphons with heavy pressure heads, the normal pressure at the invert of the deepest of these siphons being 1,000 lb. per square inch.

*Welfare*—At present about 4,900 employees are on the payroll, although at times during the construction period the number exceeded 10,000. Comfortable dwellings have been built; likewise, an up-to-date hospital, a church, and schools for both Chileans and foreigners. Continued efforts are being made by the company to make Potrerillos an attractive community for its employees.

# The Polish Enterprise

## *Anaconda Introduces American Methods in a Historic European Lead-Zinc District*

By George Sage Brooks

*President, Giesche Spolka Akcyjna*

CHRONICLERS of the latter-day Hapsburgs have related that the charming and capable Empress Maria Theresa was wont to refer to her mining province of Silesia as the fairest pearl of her necklace of Austrian possessions. She might have added it was also one of the most ancient. As an area exceedingly rich in natural resources Silesia had been recognized as an important mining district earlier than the seventh century. In fact, of the early European mining sections, it was among the first to show production in lead, silver, and *galmei* (calamine).

Upper Silesia, in the southwestern corner of the new Polish Republic, is the section wherein are located most of the mining, smelting, and manufacturing activities of the Giesche Spolka Akcyjna. Certain of the company's mines in Galicia, which lies adjoining and directly east of Silesia, are mining orebodies which were being exploited for lead and silver as early as 1415. In fact, the operations at Trzybinia are on lands which are described by the original Latin records of concessions granted by the Polish King Vladislai Jagiello to the family Trzybinia. Still more ancient is the mining history disclosed, in both Polish and Latin text, on the "Antiquity of Metal Mines," published in Warsaw in 1791. Therein are found quotations from the historians Tacitus and Ptolemy relating to those activities centering about the headwaters of the Vistula during the reign of Constantine.

American fiscal control, methods, management, and capital were the basis of Anaconda's undertaking the operations of the Giesche Spolka Akcyjna in 1926. These operations are interesting because they embrace an entire basic industry, including the mining of coal, zinc, and lead, and the concentration and smelting of their ores. Nor is this all; additional integration is secured through ownership of metal fabrication, rolling mill, and byproduct chemical plants, not to mention large timbering activities and the operation of sawmills. There are also collateral holdings in a company manufacturing explosives and paper. The largest and most modern porcelain plant in Poland, a fertilizer plant, and five brick plants complete the manufacturing units. Five large domains or estates furnish grain, milk, potatoes, and other food used in the company-owned towns near the mines and smelters. In its breadth, diversity, and completeness, therefore, the Anaconda Giesche Company is one of the most comprehensive American industrials in Europe today.

As far as we know, this group is the oldest uninterrupted basic enterprise owned and operated by an American company. In terms of individual business life, it has been carried on continuously for more than 249 years. Measured against our own historical background, these mining interests of Giesche's were started a hundred years before our War of Independence.

In taking over the properties and plants, the new management found that, in addition to modernization of equipment and technical methods, the commercial branch of the business also required complete reorganization. The merchandising, commercial procedure, and cost accounting were put upon American standards so far as was possible in European trade. The job of fitting the best of American industrial practice into the old order was an undertaking as difficult as it no doubt appeared presumptuous. Of course, many modifications were necessary to conform to the prevailing economic-social fabric of the Silesian district of the republic, as well as certain of the fixed commercial standards of Continental business.

The entire merchandising of the company is controlled through a general sales department at Katowice. Branch sales offices distribute from Warsaw, Berlin, Danzig, Vienna, Prague, and Breslau, with subagencies in London, Paris, Milan, Copenhagen, Stockholm, and Budapest. In 1928, the following materials were distributed: 133,467,000 lb. of slab zinc; 13,953,000 lb. of pig lead; 2,976,000 tons of coal; 5,347,350 zinc battery cups; 31,466,000 lb. of zinc sheets; 809,000 lb. of lead pipe; 811,000 lb. of lead shot; 5,072,000 lb of litharge and red lead; 25,000,000 bricks; 103,590 tons of 60 deg. Bé sulphuric acid; and 67,360 tons of superphosphate and mixed fertilizers.

About 22,000 employees are distributed throughout the various mines and

plants. Extensive housing, recreation, and service centers are provided for the workers. Over 100,000 people are housed by the Giesche company, which also operates its own hospital for the smelter men at Rozdzien, and, further, has the largest single interest in the Silesian industrial pension, insurance, and hospital association, which maintains nineteen hospitals throughout the district.

**Mining properties of the Giesche company**

The principal zinc mine of the company is situated at Bleischarley, adjoining the boundary between Poland and Germany. The ores consist of zinc sulphide (largely as wurtzite), galena, and zinc carbonate (locally known as *galmei*), with a combined metal content of approximately 18 per cent. The main orebody, which lies almost horizontally, is approximately 100 meters below the surface, in a dolomitic Triassic limestone. It ranges from three to six meters in thickness.

The mine is extensively developed, and is now equipped with modern hoists, underground haulage, and pumping and ventilating plants, all electrically operated. Since the American control, rationalization of the underground operations has been practiced with profit. Contract work, based upon current time studies, has shown markedly better results than the previous Silesian mining methods. Wherever possible, labor-saving mechanization has been introduced.

Matilda mine, near the town of Chrzanow, is equipped with a modern power plant, electrical hoists, and a large pumping plant. The ore consists of *galmei* (zinc carbonate) and galena, with a combined metal content of between 14 and 15 per cent. Mine-run ore hoisted here is shipped to the Bleischarley concentrator.

The eastern Bleischarley field, which is held jointly with the Hohenlohe Company, is a large area, and adjoins the main Bleischarley mine directly to the east. Development work in this section is now being carried forward to prove and delimit the orebodies, consisting of zinc sulphide, galena, *galmei*, and pyrite. The depth of this orebody is approximately the same as that of the main Bleischarley mine.

The company's coal mines hoist approximately 14,000 tons daily. This production is divided among three separate groups of mines, the first and most important of which is the Giesche group, at the southeastern edge of the city of Katowice. It consists of four main hoisting shafts, with their corresponding screening and sorting surface plants. The Cleophas group is on the northwestern edge of Katowice. There hoisting is done through two principal shafts. Auxiliary sorting and screening plants, as well as a power plant, are included in the extensive surface equipment of these properties. The Bory mines are in Galicia, not far to the northwest of the lead and zinc property of the Matilda mine. These mines produce approximately 25,000 tons of coal monthly. The accompanying sketch indicates the relative location of coal- and metal-mining units, but does not include any of the manufacturing, smelting, or concentration plants.

The company's ore is concentrated at Bleischarley. Originally, work was done in three plants near the main shaft of the Bleischarley mine, but the Anaconda staff, in carrying out its rationalization program, closed the two smaller plants. The main concentrator, with a capacity of approximately 3,000 tons of mine-run ore daily, has been completely overhauled, and an additional flotation section has been added, which is the largest in Europe. Briefly, the practice consists in deleading in the gravity section simultaneously with the production of about 40 per cent of the entire blende concentrates as gravity concentrates. The remaining 60 per cent is produced in the flotation units. The grade from the gravity section is approximately 52 per cent zinc, and from the flotation section, 58 per cent. *Galmei* and mill slime, containing about 10 per cent zinc, are shipped to the fire concentration plant for treatment by the Waelz process.

To utilize the extensive developments of *galmei* ore, the general metallurgical management, after the first examinations, decided upon the application of a fire concentration method, whereby the zinc carbonate could be converted into zinc oxide for subsequent treatment by the electrolytic process. The carrying out of this step was most important for the best utilization of the ores. Its adoption has meant an extension of the ore reserves and a reduction in the cost of producing metal from *galmei*, as heretofore carried out by the old company.

The Waelz plant, situated about one-quarter mile from the Bleischarley concentrator, has a rated capacity of 680 tons of dry ore daily. It consists of five units, each made up of Krupp revolving furnaces (40 meters long by 3½ meters in diameter), combustion chambers, sections of U-pipe dust collectors, cooling trails, exhaust fans, and standard bag-houses, using muslin bags. The collected oxide is given a subsequent roasting treatment before shipment to the electrolytic plant.

The electrolytic plant, to which the roasted zinc oxide is transported from Bleischarley by the Polish Railways, is located on the site of the Bernhardi zinc plant. It has a production of approximately 52 metric tons per day. The first cathode sheets were taken off early in November, 1928, and the construction of the entire plant, including leaching, tank room, and melting units, was completed within eleven months after breaking ground.

All the zinc sulphide ores are treated in three retort smelting plants, Uthemann, Bernhardi, and Trzybinia. The Pauls and Wilhelminia plants were closed by the American management with the modernization of the other three works. Pre-roasting is carried out in mechanical furnaces of the Wedge and Spirlet type. This is followed by sintering on Dwight & Lloyd straight-line and round machines. Byproduct sulphuric acid is made by both chamber and contact processes. Improvements have been made in the retort furnace equipment, which originally consisted of a Silesian modification of the Overpelt-Belgian reversing, recuperative, producer-gas-fired furnace. Three auxiliary potteries serve the retort plants.

The company is operating one lead-smelting unit, called the Walther Croneck plant. A second at Trzybinia has been closed in the rationalization program. The Walther Croneck plant adjoins the group of zinc-smelting plants consisting of Pauls, Uthemann, and Bernhardi. It treats lead ores from Matilda and Bleischarley, the electrolytic plant residues, and oxide roaster plant lead products. The plant consists of roasters, H. & H. pots, blast furnaces, and a Parkes refining section for silver recovery. Red lead and litharge are also manufactured here.

A new electrical power generating plant has been completed this year at the principal shaft of the Giesche coal mines. Fine coal is handled by mechanical *hanger-bahn* directly from the shafthouse to the coal bunkers. This unit has a capacity of 32,000 kw., and is equipped with modern Swiss turbo-generators built by Brown Boveri. This construction was made necessary by the new electrolytic zinc plant at Bernhardi, a short distance away.

A second power house is operating at the Cleophas coal mines; also, a third unit, using part waste-heat steam from the zinc furnaces at Bernhardi, is located at that plant. Within this group of three plants is generated sufficient power for the operation of the coal mines and zinc smelters in Silesia. The new electrical equipment at the Nickisch Schacht power station was furnished in large part by A. E. G. and Siemens-Schuckert.

Inter-plant handling is done by Giesche railroads and by Polish State narrow and standard gage lines. Giesche also operates passenger service on its own lines for miners between company villages and mines.

The manufacture of battery cups, extruded metal, rolled sheet zinc, rolled lead, shot, and special lead shapes is carried out in three separate mills at Bernhardi, Trzybinia, and Walther Croneck. At Recke Hütte the company operates a roasting plant in connection with chamber sulphuric acid plants. Here is carried on a custom roasting business, using Silesian and Spanish pyrites, as well as blende.

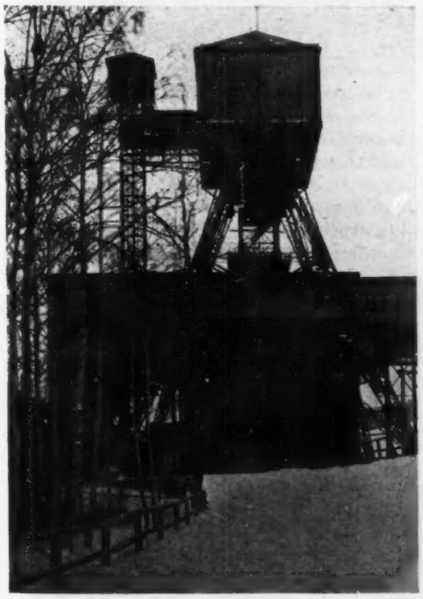
Among its other byproducts, the company produces about 22,000,000 bricks annually. These are made in five modern brick plants. One of these manufactures high-grade refractory ware for the district, supplying its own zinc plants, as well as the steel and general zinc industry of Silesia. Raw materials for the better grades of refractories are imported from Czechoslovakia and Germany. The largest plant in Poland for the production of electro and table porcelain is owned by the company and located at Katowice. Approximately 1,000 employees are on the factory payroll.

A phosphate fertilizer plant, the products of which are sold entirely within Poland, is also operated. Sulphuric acid is furnished from the company's acid plants, and crude phosphate rock is brought to Danzig by ship from North Africa and Florida.

The company has the largest single interest in the Lignoza Explosive Company, which operates two factories for the production of mine explosives and from which Giesche obtains supplies for all its mining operations. This same company is now engaged in the building of a paper plant, to work in conjunction with the explosives plant.

Giesche company's timber areas are extensive, and are exploited by the usual European method of systematic cutting and reforestation. Important holdings in sawmill operations extend from the plants of the Ober Silesian Holz Industrie at Tarnowitz, south across the Polish border into Czechoslovakia, where a sawmill is located in the Carpathian Mountains. Exploitation is also carried on north of Poznan, as well as east of Lwow, near the Russian boundary. Mine timber, as well as finished lumber, is produced.

Giesche also has under construction coal-loading plants, one at the Polish seaport of Gdynia, and another in the Free State of Danzig. These new plants are of modern quick-loading, mechanical type, and are to serve in the Giesche coal-export business on the Baltic.



**Main shaft of the Bleischarley mine**

A tripartite association is carried on between employer, workers, and the government on questions relating to industrial social problems. In this, the broad and energetic development of commerce is the aim of the Polish Ministry.

In the economic rehabilitation of a newly re-established, age-old nation, in which the Polish race is now so intensively occupied, the introduction of certain American standards may, it is sincerely hoped, prove a constructive contribution from the American republic to Poland.

# Anaconda Wire & Cable Company One of the Newest Subsidiaries

By H. Donn Keresey
*President*

ANACONDA made its initial step into the field of manufactured copper products on June 9, 1918, when the Rolling Mills Department at Great Falls, Mont., started production. This department comprised the most modern copper rolling, drawing, and stranding facilities of the time. In almost each successive year, the Rolling Mills Department made a new record production. It is interesting to note that the first substantial order received was for the electrification of the Chicago, Milwaukee & St. Paul Railroad, nearly the entire amount of copper wire and messenger cable required in that project being supplied by the Great Falls mill.

In February, 1922, Anaconda's purchase of The American Brass Company brought it into the leadership in the wire manufacturing field. Soon it became apparent that the rapidly growing requirements of the Middle West demanded the erection of wire facilities in that territory, and in November, 1923, work was begun on the new wire mill of The American Brass Company at Kenosha, Wis., with possibly the largest wire-drawing capacity of any in the world today. The position of the company in the wire field was now substantially established, but the management, persistently following its policy of "Mine to Consumer," realized that the logical final step should be taken—that is, manufacture of insulated wire. And so in February, 1929, a subsidiary known as Anaconda Wire & Cable Company was created, which absorbed the mills at Great Falls and Kenosha. This was followed by certain other steps. Cordial relations had been established with two of the largest manufacturers of insulated wire products in the Middle West—namely the Inland Wire & Cable Company and the Maring Wire Company. Inland was by far the largest consumer of copper rods in the country and Maring was not far behind. Incidentally, Inland was one of the largest manufacturers of weatherproof wire, and was a consolidation of the Illinois Wire & Cable Company and the Chicago Insulated Wire & Manufacturing Company, both of Sycamore, Ill. Quite naturally, therefore, considering its entrance into the insulated wire field, Anaconda's first thought was to the long-established relations with these two customers.

Late in February, 1929, Anaconda Wire & Cable Company acquired the assets of the Inland company, and in April of the same year the business of the Tubular Woven Fabric Company, of Pawtucket, R. I., a well-known manufacturer of rubber insulated wire and kindred products. In the meantime, negotiations had been proceeding with the officials of the Maring Wire Company, with plants at Anderson, Ind., and Muskegon, Mich., which were successfully completed late in April, 1929.

Anaconda Wire & Cable now had complete facilities for the manufacture of all bare and insulated wire products with the exception of underground power cable. This was added when the negotiations for the Hastings-on-Hudson plant of the Hastings Wire & Cable Company, a subsidiary of American Brass, were closed in May, 1929.

As a result of these consolidations the plants comprising the units of Anaconda Wire & Cable Company, and their products, are now as follows:

Great Falls, Mont.: Rods, bare wire, and cable.

Kenosha, Wis.: Bare and tinned wire and cable.

Sycamore, Ill.: Bare and weatherproof wire and cable; office, annunciator, and magnet wire.

Muskegon, Mich., and Anderson, Ind.: Bare and magnet wire.

Hastings-on-Hudson, N. Y.: Rods, bare and tinned wire and cable; weatherproof wire and cable; varnished cambric and lead-covered paper-insulated power cable.

Pawtucket, R. I.: Rubber-covered wire and allied products.

In addition to these plants, Anaconda Wire & Cable Company acts as selling agent for the following mills of The American Brass Company:

Detroit, Mich.: Rods, bare and tinned copper wire and cable.

Ansonia, Conn.: Rods, bare and tinned copper wire and cable; magnet wire.

Waterbury, Conn.: Bare and weatherproof wire and cable; office and annunciator wire.

# How Anaconda Applies Electricity in Mining and Metallurgy

By C. D. WOODWARD

IN RECENT YEARS nearly all schemes for improving production in industry involve the use of electricity, because employment of this agent is usually the most promising means for simplifying and cheapening methods of operation. Starting at a time when the electric motor was a curiosity and before it had passed through its probation period, the mining industry recognized that this agency would become a dominant factor in solving many of its problems, and that its use might be essential to its existence; therefore, within the last two decades electricity has revolutionized the non-ferrous metal industry, because, in mining, it has practically replaced steam and animal power and in metallurgy has made possible the reclaiming of metals from complex ores which previously had little or no commercial value.

The use of electricity in the mining industry has become almost universal. Long ago the mine candle was generally replaced by the electric light and the mine horse or mule by the electric locomotive; steam and air were discarded for motor-driven pumps, and the steam hoisting engine, shovel, and churn drill are giving way to the electric drive. The movement of air for mine ventilation is now accomplished more efficiently by electric motor-driven fans; the air compressor has been electrified; and the steam locomotive that hauled the ore from the mines to the reduction works has been replaced by the electric motor.

Deeper mines brought about the development of electric signals and the use of telephones to dispatch the mine cages through the shafts, their safe and accurate operation being guarded by electric controllers. Finally, electricity is used for heat-treating the drill steel and to repair and reclaim, by the electric welding process, many mining equipment parts.

*Electric Underground Haulage*—When the distances were short between the working face and skip pockets, ore and waste were trammed by manual labor. Later, animal power was introduced, and in some mines replaced by compressed-air locomotives. With the enormous quantities of material handled underground every day, it was obvious that the electric mine locomotive could be introduced and would effect a tremendous saving. With this end in view, the trolley locomotive practically replaced its predecessors, but later, to find a machine better adapted for under-

ground haulage, storage-battery locomotives were installed, and they have now become a standard with this company, and eventually this type of machine will retire all others.

Two sizes of storage-battery locomotives are in use: a 5½-ton unit for main haulage and a 1½-ton unit to replace man and animal power in the older workings, where bad ground, poor track conditions, small drifts, and short radius curves prevent the use of the larger units. Both types were developed for the severe underground conditions. The larger is equipped with 105 cells of Edison A-5 batteries. It has a drawbar rating of 1,500 lb. at 3½ miles per hour. An average mine train for this particular unit consists of ten loaded cars, giving a trailing load of approximately twenty tons. The smaller locomotives, or Little Trammers, are of compact design and constructed so they may be placed on a mine cage without dismantling and transferred to different levels. Its average load consists of four loaded cars, and its batteries, in common with those of the larger type machine, are of the Edison make.

Power for battery charging is furnished from 250-volt automatic substations at convenient points underground, and as the batteries are not removed from the locomotive for charging, the mine levels are furnished with dead-front charging panels, with necessary interlocks, which make their operation simple for the motorman. The locomotives are charged between shifts, or in the valley of the power loads, so no power peaks are established and the resulting economy is obvious.

*Electric Underground Pumping*—In the early days steam and compressed air were used to operate the pumps in the Butte mines. With the former inefficient method of long steam lines down mine shafts, and pumps exhausting into sumps, large quantities of heat had to be dissipated in the mine workings and taken care of by natural ventilation. Compressed-air pumps required heat or steam to keep them from freezing, and their performance was likewise unsatisfactory as well as inefficient. As the mine became deeper, greater pumping facilities were essential. The steam and compressed-air pumps were replaced by the larger and more economical electric motor-driven pumps. The standard Anaconda mine pumps, of which 26 are in operation, each have a capacity of 500 g.p.m. under 1,200-lb. pressure. They are directly driven by two induction type, 150-hp., 2,200-volt motors, through a single reduction of herringbone gears and controlled by dead-front safety switchboards. In addition to the standard pumps, many others are equipped with motors ranging from 50 to 150 hp.

*Hoisting*—All deep metal mines require efficient and reliable hoisting equipment, and electrical energy is used for practically all hoisting operations in the Butte district. Years ago, general progress dictated the substitution for steam hoists of engines operated by compressed air. This change permitted the use of cheap hydro-electric power and resulted in many hoisting economies. In later years, as hoisting equipment became inadequate for increased mining requirements or new developments, new electric-driven auxiliary and main hoists were installed. The new electric hoists in sizes up to 500 hp. are furnished with alternating-current equipment. For the larger hoists, direct-current motors coupled directly to the hoisting mechanism and serviced by motor-generator sets, incorporating the Ward Leonard system of control, are employed. Where widely fluctuating loads on the power system are objectionable, the motor-generator set is furnished with a balance wheel arranged so as to absorb or give up its energy at proper intervals in relationship to the hoisting cycle. This scheme, known as the Ilgner system, is used on a few of the large hoists in the Butte district.

As the developments of the power company furnishing the current became more extensive, fluctuating hoisting loads were less serious, and as a consequence the motor-generator sets

for the new large hoists of the Anaconda company are now driven by synchronous motors and require no balance wheel. The new large hoists embody the latest engineering in their electrical and mechanical designs. By virtue of the Ward Leonard control, regenerative braking is incorporated, and this, being safe, readily lends itself to heavy-duty hoisting with automatic or semi-automatic control, resulting in the most economical hoisting equipment known.

*Electric Shovels* — In the Chilean open-cut mines, some twenty electrically operated shovels are now in successful operation. These shovels have replaced the steam shovel which was formerly used, but before the present type was installed it was preceded by an electric shovel driven with alternating-current motors. The change from steam to electrically driven shovels naturally resulted in greater operating economies, which were further increased by the introduction of another type. This type of shovel is mounted on caterpillar tracks having numerous special features to meet particular mining requirements.

To withstand the rough handling of Chilean operation, it follows naturally that they must be strongly constructed and foolproof, aside from incorporating many other features required by the peculiar climatic and local conditions. The later shovels are equipped with a motor-generator set with a motor wound to operate on a 3-phase, 50-cycle, 5,000-volt supply. This motor drives three generators and an exciter for the operation of the 230-volt direct-current motors actuating the various shovel motions, these in turn being controlled by the Ward Leonard system.

Many churn drills are used in conjunction with the shovel operations. These machines are similar to the standard steam-driven drill, except that they are diven by electric motors.

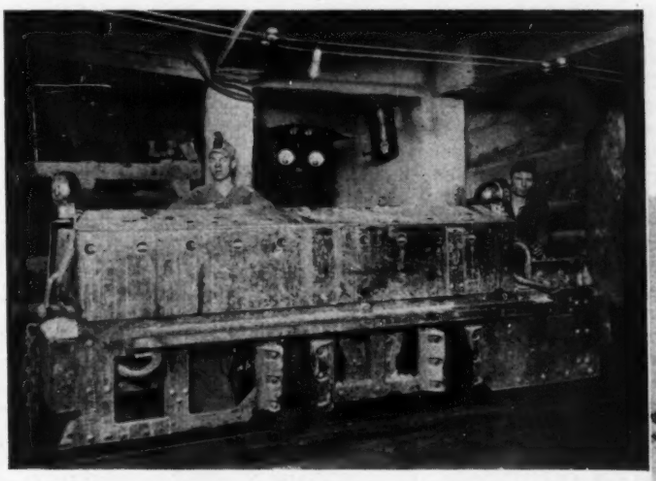
*Ventilation in Mines*—It has long been recognized that an abundance of fresh air supplied to the mine workings increases the efficiency of the human body and mind, as well as prolonging the life of the timber and producing other beneficial results. Since it is necessary to bring about normal working conditions, artificial ventilation is demanded. This has been accomplished by large exhaust fans placed over shafts at the surface and supplemented in many instances by booster fans underground. All such fans are driven with electric motors, ranging from 50 to 300 hp. Aside from this, to secure a flow of air through lower levels and to divert the air into active sections of the mines, hundreds of smaller fans are in service, which are directly connected to fire- and drip-proof motors designed for the purpose.

*Locomotives*—The Anaconda Copper Mining Company and its associate companies transfer the ore between their mines and reduction works over railroads which are electrified. The problem of a haulage unit embodying the economy and flexibility of an electrical operation that will function, not only on permanent trackages of the reduction works but also on the rough tracks of mine and tailing dumps, has been solved by the use of several types of electric locomotives. For use around mines and smelters, at least six types of heavy locomotives have been designed and are in successful operation. Perhaps the most interesting are those which were built for the Chile Exploration Company. These embody the flexibility of the steam and the economy of the electric locomotive. They were built to function on permanent tracks between the mines and reduction works, as well as on the rough tracks of the mine benches where the use of trolley wire or third rail is not feasible. Since fourteen of these locomotives are doing the former work of 25 steam locomotives, a description would be interesting, but space limitation permits only an outline of the essential features.

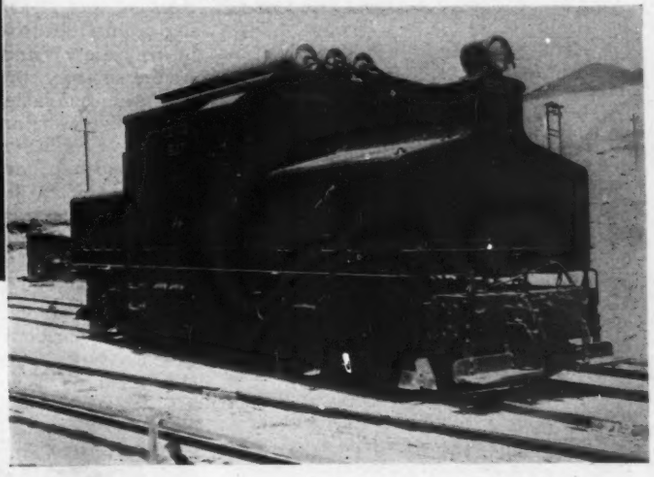
Each locomotive weighs approximately 80 tons, equipped with eight drivers, 36 in. in diameter, geared to four 165-hp. motors mounted in trucks of articulated construction. The over-all length of the locomotive inside of knuckles is 36 ft., with an approximate over-all width of 11½ ft. and a height of 14 ft. When the locomotive is operated on a 650-volt third rail, the rated draw-bar pull is 27,000 lb. at about 9 m.p.h. These machines are equipped with 120 cells of lead-type storage batteries having a capacity of 160 kw.-hr., and when supplied with power from the storage battery and operating on the mine benches, each has a speed of about 5 m.p.h. with a 10,000-lb. tractive effort.

Battery charging is accomplished by a motor-generator set on the locomotive, serviced from the third rail while the locomotive is operating between the mine and reduction works. Aside from this, each locomotive is provided with control features for tandem operation as well as with air compressors for automatic and straight air braking and the actuation of third-rail collectors. Also, there are suitable receivers with control equipment for car-dumping requirements.

*Air Compressors*—Many hoisting engines in the Butte district are operated by compressed air, as has already been remarked. The air for the engines, as well as for other mining operations, is supplied from central plants strategically located in reference to the mines. Compressed air is distributed to the different mines through about 16 miles of surface pipes, and pressure is maintained at 90 lb. by a hydrostatic system. Smaller mines in the outlying districts are served by individual compressor plants. A standard air compressor having a capacity of 7,500 cu.ft. per minute, and driven by synchronous motors, supplies most of the compressed air. All of the air compressors are electrically driven, and approximately 18,000 hp. is used for supplying the mine drills and some of the hoisting operations.



**Standard 5½-ton storage-battery mining locomotive**



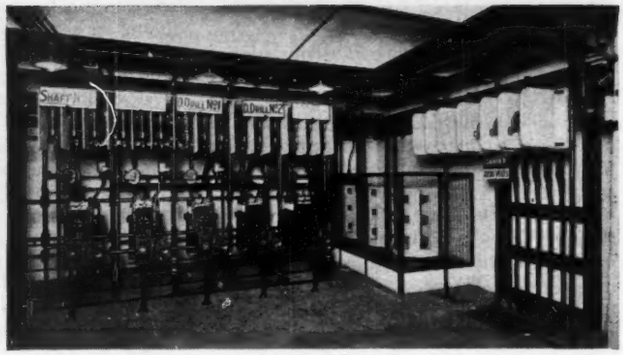
**Combination third-rail and storage-battery 76-ton locomotive**

*Electric Signals*—Another important essential that contributes to safe mine operations is the reliable, simple, and flexible signal system for the dispatching of mine cages through the shafts. To meet severe mining conditions, this company has developed equipment and a system of signals that have become practically universal in all deep metal mines. In general, two independent signal systems are installed in each hoisting compartment. These are supplemented by another system which sounds a buzzer at each level in the mine for the purpose of calling a station tender. One of the signal systems is known as the A.C. or Station Bells, and the other is known as the D.C. or Shaft Bells. As the name implies, the A.C. system derives its energy from an alternating-current supply, and this system is normally used to dispatch skips and cages at all times, except when power or equipment failures render it inoperative.

> Mining and metallurgy are immense industries, and today electricity is the essential of their existence. To turn the wheels in the many steps, or isolate the molecules, requires practically every known type of electrical device or apparatus. As time goes on, with its rapidly increasing developments, and necessity of further economies, electricity will demonstrate that it is indispensable and that it constitutes the real solution of a large proportion of the industry's problems.

The D.C. system is energized from storage batteries and primarily maintained as an emergency for the A.C. system. In addition, it is useful in signaling between levels, as the pull switches at each station are furnished with ropes extending down the shaft to the next station. This arrangement facilitates the transmission of signals while shafts are being repaired and inspected. In addition to the signal systems, there is a mine telephone on each level, each connecting with an instrument on the engineer's platform.

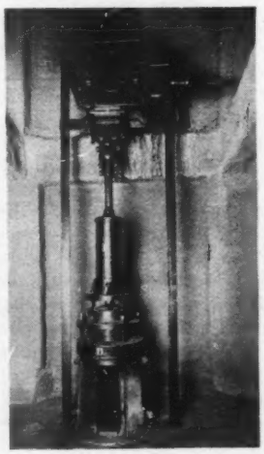
*Power Distribution Underground*—Since thousands of electrical horsepower are used for electrical apparatus underground in the Butte mines, and as continuity of service, reliability, and safety are paramount, the energy is distributed over duplicate armored cables placed in diamond-drill holes or in fireproof shaft compartments extending to central distributing points underground. At points of distribution the potential is stepped down to lower voltages for small motors and lighting installations. Early underground electrical installations, using improper cables, were sometimes the cause of accidents, and, likewise, fires resulted because such cables were improperly insulated and supported, besides lacking sufficient strength to support their weight or resist impact when struck by falling material. In general, present practice prescribes main cables insulated with varnished cambric and operated at half their rated voltage. These cables are lead-jacketed, being overlaid with a protective armor coat of lead-covered, galvanized steel wires. When they are so constructed, such cables have sufficient strength to support themselves in lengths of from 500 to 1,000 ft. when hung vertically in diamond-drill holes, and, further, they are not easily damaged by falling material when placed in shaft compartments.



**Surface terminal of mine cables, showing switch and current-limiting reactors**

As it is frequently necessary to make cable half-taps, special junction boxes are used which provide means for sectionalizing, as well as sealing the cable from moisture and supporting the conductors and armored strands. On levels where cables are tapped, there are fireproof rooms, furnished with latticed iron doors, and at the central distributing points, dead-front switchboards (to insure safety) and lightning arresters. At the surface, current-limiting reactors are used on cables transmitting large blocks of power, and for parallel operation or duplicate service they are equipped with overload and reverse power relays to give selective isolation in the event of insulation failure.

*Electricity in Metallurgy*—In general, through the economic and flexible transmission of power, the metallurgical industry has been expanded, production increased, efficiency improved, and new methods have been developed for the reduction of ores and the refining of metals. Anaconda and its associated companies use for this purpose approximately 350,000 hp., of which about one-half is generated in its steam power plants.



**Cable support and half-tap diamond drill-hole installation underground**

*Electrical Deposition*—For the deposition of electrolytic zinc from sulphate solutions in the two Montana zinc plants of the company, about 100,000 electric horsepower is used. The power is purchased for both plants in the form of 3-phase current and delivered at 100,000 volts. From that figure, the potential is transformed in one plant to 400 volts, and in the other to 6,600 volts.

Inasmuch as direct current at approximately 550 volts is required for the electrolytic zinc cells, the power is purchased in the form of high-tension alternating current, and the transformation and conversion from alternating to direct current is made in one substation by synchronous rotary converters operated in conjunction with synchronous booster sets, and in the other substation by three-piece synchronous motor-generator sets.

The electrolytic deposition of copper from leaching plant solution is practiced on a large scale at Chuquicamata and Potrerillos, Chile. At the former plant about 80,000 hp. is required, and the latter uses 15,000 hp. The direct current for the electrolytic departments is transformed from transmission potentials of 100,000 and 80,000 volts, respectively, and converted from alternating current, by means of synchronous booster rotaries and motor-generator sets, to potentials below 400 volts, direct current. In the United States, Anaconda operates two electrolytic copper refineries, the necessary direct current being supplied at 150 to 230 volts from motor-generator sets.

Obviously, both synchronous converters and motor-generator sets are used to supply direct current to electrolytic circuits. Detail costs and estimates indicate that motor-generator sets cost slightly less per kilowatt installed than synchronous converters, which have slightly higher efficiencies. But as synchronous rotary converters require greater skill in operation, and their maintenance cost is much more than that of motor-generator sets, this factor offsets the power saving effected by the converters' higher efficiency. Then, too, if power factor correction is re-

quired, this can be accomplished better with synchronous motor-generator sets; and, finally, direct-current voltage regulation for electrolytic circuits requires careful consideration, because at times radical changes in the voltages are necessary, and this feature may be accomplished better by the motor-generator set.

From the foregoing it is apparent that almost every electrolytic plant must be treated as a separate problem and that it is not possible to make a choice between the synchronous converters and motor-generator sets until all phases of the problem are carefully analyzed.

*Motor Applications* — Thousands of horsepower in electric motors are used for driving, crushing, concentrating, roasting, and converting machinery. For necessary power factor correction, the synchronous motor has been applied wherever possible. For the crushers, and many applications in the concentrators, strongly constructed mill-type induction and slipring motors are installed; and for roaster furnaces or for machinery operating in other dusty places, entirely inclosed, force-ventilated induction type motors are applied. For copper converters and cranes the direct-current mill type is best. In the event of power failure, resulting in the loss of converter air supply, storage batteries can be used and arranged to lower the converters automatically, thus removing the possibility of the tuyères becoming filled with molten copper.

Since safety in the handling of electrical equipment is one of the chief considerations, and as most of the motors installed are operated by unskilled labor, automatic protection and push-button control are necessary. This is best illustrated by the synchronous motors driving Marcy mills in the Andes plant at Potrerillos. For this application, Westinghouse so-called internal clutch-type motors, each rated at 400 hp., are used. Automatic, remote control stations for starting each motor and engaging the clutch are located on platforms above the mills. When the motor has been brought up to full speed by push-button control, another button is pressed, starting a small motor that actuates another mechanism which gradually applies current to the clutch windings and causes the motor to accelerate the ball mill smoothly.

*Cottrell Precipitators* — Another important use for electricity in metallurgical plants is the application of high voltage, direct current in Cottrell treaters to recover dust from flue gases. In general, the substation for the treaters consists of equipment to step up alternating current to 50 or 60 kilovolts by transformers serviced from single-phase generators. These latter machines are driven by synchronous motors, which likewise drive the mechanism for rectifying the high-voltage alternating current to direct current. With such an arrangement, it is easy to obtain a uniform voltage gradient for regulating purposes. Some installations have been made without the single-phase generators, but regulation is then acquired by changing taps on the transformer windings or by using an induction regulator.

Since the first Cottrell treater units were developed at the Anaconda Reduction Works, various plate-, tube-, and mesh-type treater units have been built, and substations using vacuum-tube rectifiers or Kenetrons, as well as synchronous motor-driven mechanical rectifiers, have been installed and are in successful operation.

*Power Transmission and Distribution* —Power for the Chile and Andes developments in South America is generated from steam plants on the seacoast. These plants, the largest on the west coast of South America, use oil for fuel. The Chile plant delivers over three copper circuits, supported on steel towers, 80,000 kw. at 100,000 volts to the metallurgical works, 87 miles distant. The Andes plant furnishes 23,000 kw. to the mines and smelter at 80,000 volts over two copper circuits 78 miles long which are supported on double-circuit steel towers. Since reliability and continuity of service are essential, and as the rough character of country, with its unusual climatic conditions, imposes severe construction and operating conditions, it follows that the design and construction of the transmission lines embody many details which are peculiar to this particular locality.

The electrical distribution system for a metallurgical reduction plant presents nothing unusual except that it must be arranged so that failure of any circuit results in a minimum of interference with the plant's production. In the main, this requires a grouping of electrical circuits to parallel the flow of ore through any division of the plant, and a brief description of such a system as used by the Andes company serves to illustrate the application.

The concentrator is divided into five sections, and a substation is provided for each section. Individual power circuits are brought to each substation, in which is located the remote-control equipment for each motor in the section, including the bin gate, apron feeder, ball mill, classifier, and flotation pump motors. Naturally, the failure of one circuit affects one section only, and the four others continue to operate. The same methods are applied likewise to the coarse-crushing and the fine-crushing plants.

Mining and metallurgy are immense industries, and today electricity is the essential of their existence. To turn the wheels in the many steps, or isolate the molecules, requires practically every known type of electrical device or apparatus. As time goes on, with its rapidly increasing developments, and necessity of further economies, electricity will demonstrate that it is indispensable and that it constitutes the real solution of a large proportion of the industry's problems.

# Half a Million Tons of Coal Produced Annually by Anaconda

By F. W. C. WHYTE
*General Manager, Coal Department*

THE COAL PROPERTIES of the Anaconda Copper Mining Company are in Lincoln County, Wyo., and in Carbon and Cascade counties, Mont. The property in Wyoming is adjacent to the town of Diamondville and is operated by the Diamond Coal & Coke Company, a subsidiary. Two of the three mines on the property are in operation at present, the coal being hoisted from slopes which vary from 13 to 20 deg. The main vein, generally known as the "Kemmerer" vein, is normally 14 ft. thick and is a high-grade semi-bituminous coal, especially adapted to the use it is put to by the company—namely, smelting. Under part of the property there is also a 7-ft. vein of high-grade coal, lying about 50 ft. above the main vein. This smaller vein is being worked in two of the mines. In addition to the coal used by the parent company, a large amount is used by the Oregon Short Line Railroad and a considerable quantity is sold commercially. All three mines are fully equipped with tipples and electric pumping, the electric power being furnished by a large power plant at No. 2 Mine.

The property in Carbon County, Mont., is at Washoe, and is operated by a rock slope, fully 600 ft. long, driven on a 30-deg. pitch. The vein being worked is the "Bearcreek" No. 2 vein and is normally from 7 to 8 ft. thick. It is a high-grade semi-bituminous coal and is sold commercially throughout Montana, North Dakota, and eastern Washington. In addition to the No. 2 vein at least three other veins of workable coal lie under the property. The mine is fully equipped with electric hoist, shaking screens, picking tables, electric pumps, and underground haulage, and is in position to ship a first-class commercial coal. All the larger sizes are sold commercially, whereas most of the small coal is shipped to the company at Butte and Anaconda.

The properties in Cascade County, Mont., have not been worked for some time now, owing to the use of oil and gas by the smelter at Great Falls.

Since 1896, when the company first began to produce coal for its own use, the coal properties have produced about 27,000,000 tons of coal, a large part of which has been used in mining and smelting. At present the coal mines are producing 500,000 tons annually.

# Industrial Relations and Safety Work

By JOHN L. BOARDMAN

THE varied industrial relations and safety work of the Anaconda company at Butte, consist chiefly of the satisfactory adjustment of questions concerning wages and working conditions, prevention of accidents, and properly caring for injured employees. Living quarters, stores, and amusement centers are privately owned, without competition from the company.

Of primary importance to the employee are satisfactory wages and working conditions, so the policy of the company has been at all times to work out a satisfactory adjustment or settlement. Therefore, a department has been created, presided over by the Commissioner of Labor, whose duties are to keep in close contact with all of the employees, so that all grievances or misunderstandings of any kind may be given a hearing and a satisfactory adjustment made. The office of the Labor Commissioner is maintained in the downtown section of the city, away from the executive offices, where employees may seek a conference with the Labor Commissioner at any time without coming in contact with any of the other company officials. Anaconda is proud of the fact that during the many years of its existence and operation in the Butte district it has never had any serious trouble with its employees, the few difficulties that have arisen from time to time having been adjusted satisfactorily to the employees and to the company. The company is also proud of the fact that its employees have at all times been fair and reasonable. The Labor Commissioner also handles the hospital contracts to see that employees are given adequate treatment. A contract with each of the two larger hospitals is also maintained by the company, which deducts $1 per month from the pay of each employee and pays this amount over to the hospital. For this fee the workman receives medical and surgical treatment for all illness suffered during his employment, except that directly resulting from intoxication or venereal disease, and surgical and medical attention for all injuries caused by and arising out of his employment.

Provisions made for the care of injured men are somewhat elaborate. All injury cases are sent to the hospital unless they are so slight that the patient refuses to go. An attempt is made to give first-aid treatment before removal from the scene of the accident.

Accident-prevention work is carried on through the Bureau of Safety, made up of the officers, operating heads of departments, and mine foremen. The work of this Bureau comes under the following branches:

1. Operating requirements: safe mining methods and safeguards.

2. Inspection service: to see that rules under No. 1 are complied with.

3. Accident reporting and statistics: all accidents are reported to the Bureau of Safety, the Claim Department, State Compensation Board, and Bureau of Mines.

4. Educational accident prevention propaganda: monthly safety magazine, safety bulletins, National Safety Council, and related activities.

5. Mine rescue stations and equipment.

6. Mine rescue and first-aid training, including annual first-aid contests.

Under this plan of organization for accident prevention, the mine superintendent and foreman, functioning as members of the Bureau of Safety, decide upon the safe method of developing and working the different parts of the mine, and the method of timbering, blasting, and transportation. They require grizzlies on all chutes, guard rails around all openings, safe ladderways, safe and orderly sills, guarded trolley wires, grounded electrical apparatus, safe track and rolling stock, safe conduct of mining work, such as barring down or timbering loose ground, and all other things which tend to reduce accidents.

Each mine employs a safety inspector, who travels continually through the mine, reporting to the mine foreman and to the chairman of the Bureau of Safety all failures to comply with the safety rules and requirements. He keeps the mine exits in safe and passable condition, takes care of the ventilation and distribution of fresh air to the working faces, and supervises sprinkling to keep down dust. He is also responsible for seeing that a proper number of men are trained in mine rescue and first-aid work, as well as reporting to the Bureau of Safety on each accident and making a semi-monthly report of inspections.

The office staff of the Bureau of Safety classifies all accidents by causes, keeps a record of the length of disability, distributes copies of accident reports to the Claim Department of the U. S. Bureau of Mines, furnishes copies of the inspection reports to the vice-president, general manager, general superintendent, and assistant general superintendents, and makes a monthly report of accidents to the manager of mines. The Bureau of Safety also edits and publishes *The Anode*, a monthly safety magazine of sixteen pages, distributed free to all employees. It compiles and publishes safety rule books, designs and distributes safety bulletins and posters, and selects and distributes the posters of the National Safety Council.

As a means for keeping up interest in first-aid and accident prevention, the Bureau of Safety annually conducts a first-aid contest, in which the two teams receiving the highest score are matched against the two winning teams from each of the metallurgical works in Montana, to compete for the honor of representing the Montana operations of the company in the International First-Aid Contest. During the last five years, the winning team in this inter-departmental contest has taken three first places, one second place, and one third place in the International Contest.

Each year the company, in co-operation with other mining companies of the district, holds the Butte Miners' Safety-First Field Day. The program of this giant picnic—usually 25,000 persons attend—includes a great variety of outdoor sports, the first-aid contest, basket picnic lunch, music, and dancing.

Under the direction of the Bureau of Safety two mine rescue stations are maintained. These are equipped with 85 sets of two-hour type oxygen breathing apparatus and a rescue truck to carry this apparatus to the mine in which it is being used.

Anaconda was one of the first of Western metal mining companies to take up seriously the work of accident prevention, organizing its Bureau of Safety early in 1914, and was the first of two companies in the United States, the other being the New Jersey Zinc Company, of Franklin, N. J., to purchase and install oxygen breathing apparatus for use in connection with fighting mine fires. The company has kept pace with the development of this type of equipment, and now operates two of the finest privately owned rescue stations to be found in the country.

Mine rescue and first-aid training is done by the Bureau of Mines rescue car crews when they are available. An attempt is made to have all mine rescue men review or take additional training twice each year, and when the car is not available, both this training and the first-aid training are given by the crews of the company rescue stations. As each mine is required to maintain a minimum crew of ten trained mine rescue men, this means that from 300 to 500 men must be given the practice each year, and a sufficient amount of new training done to keep the force up to the required strength.

Workmen's compensation is handled by a separate department independently of the Bureau of Safety and operating department. The company operates under the Montana Plan No. 1, or self-insurer plan. This arrangement eliminates all lost motion and expense in the matter of the injured employee getting the compensation payments due.

# The Market Report

## Copper and Lead Demand Improves; Prices for All Metals Firm

*New York, Aug. 21, 1929*—An improved demand for copper and lead has been manifest in the week ending today, lead inquiries picking up, especially in the last day or two, with the improvement in the price abroad. No particular interest, however, has been exhibited in tin or silver, and zinc has been exceedingly dull, though the prices on these metals have been well maintained. The stock market has shown exceptional strength, and other signs also point to active industrial conditions in the fall and winter, and an increasing consumption of metals.

### Daily Prices of Metals

| Aug. | Electrolytic Copper | Straits Tin | Lead | | Zinc |
|---|---|---|---|---|---|
| | Refinery | New York | New York | St. Louis | St. Louis |
| 15 | 17.775 | 46.50 | 6.75 | 6.55@6.575 | 6.80 |
| 16 | 17.775 | 46.625 | 6.75 | 6.55@6.575 | 6.80 |
| 17 | 17.775 | No Market | 6.75 | 6.55 | 6.80 |
| 19 | 17.775 | 46.875 | 6.75 | 6.55 | 6.80 |
| 20 | 17.775 | 46.50 | 6.75 | 6.55 | 6.80 |
| 21 | 17.775 | 46.375 | 6.75 | 6.55 | 6.80 |
| | 17.775 | 46.575 | 6.750 | 6.554 | 6.800 |

Average prices for calendar week ending Aug. 17, 1929, are: Copper, 17.775; Straits tin, 46.450; New York lead, 6.750; St. Louis lead, 6.554; zinc, 6.800; and silver, 52.563.

**The above quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices are in cents per pound.**

**Copper, lead, and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.**

**In the trade, copper prices usually are quoted on a delivered basis; that is, delivered at consumer's plant. As delivery and interest charges vary with the destination, the figures shown above are net prices at refineries on the Atlantic seaboard. Delivered prices in New England average 0.225c. per pound above those quoted.**

**Quotations for copper are for the ordinary forms of wire bars and ingot bars. For ingots an extra 0.05c. per pound is charged; for slabs, 0.075c.; and for cakes, 0.125c. up, depending on weight. Cathodes are sold at a discount of 0.125c. per pound.**

**Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight rate between the two points.**

**Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.**

### London

| Aug. | Copper | | | Tin | | Lead | | Zinc | |
|---|---|---|---|---|---|---|---|---|---|
| | Standard | | Electrolytic bid | Spot | 3M | Spot | 3M | Spot | 3M |
| | Spot | 3M | | | | | | | |
| 15 | 73½ | 74⅜ | 84¼ | 208½ | 212½ | 23 1/16 | 23⅛ | 24⅜ | 25 1/16 |
| 16 | 73¾ | 74⅝ | 84¼ | 209 | 213⅛ | 22 15/16 | 23¼ | 24 13/16 | 25 1/16 |
| 19 | 74⅛ | 74¾ | 84¼ | 210½ | 214⅜ | 22 13/16 | 23⅛ | 25 1/16 | 25 3/16 |
| 20 | 74 | 74 11/16 | 84¼ | 209⅞ | 213¾ | 22⅞ | 23 3/16 | 25 1/16 | 25¼ |
| 21 | 73¾ | 74½ | 84¼ | 208⅜ | 212¼ | 23 | 23¼ | 24 15/16 | 25 3/16 |

**The above table gives the closing quotations on the London Metal Exchange. All prices in pounds sterling per ton of 2,240 lb.**

### Silver, Gold, and Sterling Exchange

| Aug. | Sterling Exchange "Checks" | Silver | | Gold London | Aug. | Sterling Exchange "Checks" | Silver | | Gold London |
|---|---|---|---|---|---|---|---|---|---|
| | | New York | London | | | | New York | London | |
| 15 | 4.84⅜ | 52½ | 24¼ | 84s11½d | 19 | 4.84¼ | 52⅝ | 24 5/16 | 84s11½d |
| 16 | 4.84 5/16 | 52¾ | 24 5/16 | 84s11½d | 20 | 4.84 5/16 | 52⅝ | 24 5/16 | 84s11½d |
| 17 | 4.84¼ | 52⅝ | 24¼ | ..... | 21 | 4.84 5/16 | 52⅝ | 24 5/16 | 84s11½d |

Average: Silver, 52.625c.; Sterling Exchange, 484.302c.

**New York quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, basis 925 fine. Sterling quotations represent the demand market in the forenoon. Cables command one-half cent premium.**

### Only Near-by Copper Wanted

Practically all sellers report more interest in copper, though the total domestic sales of approximately 14,000 tons do not indicate an exuberant buying wave. Even though the middle of August has passed, more than half of the sales have been for shipment this month, and practically all the rest for September delivery, indicating the continued practice of consumers to buy from hand to mouth. Brass manufacturers, as well as wire and cable concerns, report considerably improved business, but it would seem that they are not buying the copper to cover the orders they are placing on their books. The price has been uniformly 18c. for Connecticut delivery basis, and 18⅛c. in the Middle West.

The foreign demand has been very satisfactory, especially that in England. Total sales so far in August approximate the total for the entire month of July. The price set by Copper Exporters, Inc., remains at 18.30c., c.i.f. European ports.

World production of copper in July was 174,547 tons, against 174,586 tons in June and 196,820 tons in April, the peak month, according to the A.B.M.S. statistics. The daily rate for July was 5,631 tons, against 5,820 tons in June and 6,561 tons in April. Production for the January-July period was 1,277,207 tons, comparing with 1,057,439 tons in the corresponding period last year, an increase of 20.8 per cent.

### Lead More Active

Somewhat more than an average week's business was booked in lead, with buying fairly well distributed among the important consuming industries. A feature in the situation was the persistent call for near-by lead, indicating that consumers are in urgent need of the metal. A good part of the tonnage sold was placed in the last two days, much of the buying being inspired by higher lead quotations from London. It is felt that consumers still have to purchase some August lead, and with a good part of September's wants yet to be provided for, the position of the market appears to favor sellers.

The New York price held at 6.75c. throughout the period. In the Middle West the ruling quotation was 6.55c., although two transactions did go through on the basis of 6.60c. for October shipment, early in the week.

Lead statistics for July show output of crude of 83,990 tons, against 83,246 tons in June. Stocks of refined, including antimonial lead, in hands of producers on Aug. 1 were placed at 72,873 tons, a gain of 6,614 tons over the previous month. Domestic shipments in July, according to the A.B.M.S. statistics, totaled 62,802 tons, against 57,715 tons in June.

## Zinc Neglected at 6.80c.

Though zinc sales last week were not particularly large, they have been markedly less this week. Producers are uniformly holding at 6.80c. in spite of efforts of consumers to depress the quotation. However, if any 6¾c. metal is available, it is probable that it is a small lot offered by others than producers. In the present condition of the market, with consumers evidently underbought, sellers would seem well advised in holding their quotations firm until demand reasserts itself. High-grade zinc is going out well on contracts, at the usual differential of 1 to 1¼c. above St. Louis prices of Prime Western, for Eastern deliveries, but the open market prices of spot carloads are purely nominal at 8@8¼c.

## Tin Quiet

Price changes in tin were relatively insignificant, with trading rather slow so far as consumers were concerned. Speculative activity in London has been restricted of late, and the market appears to be marking time. Spot Straits has been in somewhat better demand than forward, with a consequent narrowing of the contango.

## Silver Inactive But Steady

For the last week the silver market has continued quiet but steady, London prices being 24¼d. and 24⅝d. and New York ranging from 52½c. to 52¾c., with 52⅞c. quoted on the last four days. The Chinese-Russian situation has not caused any advance in price, and on the other hand support from the Indian bazaars has prevented any decline.

*Mexican Dollars* (old Mexican pesos): Aug. 15th, 39⅞c.; 16th, 40c.; 17th, 19th, 20th, and 21st, 39⅞c.

## Foreign Exchange

Closing cable quotations on the leading Continental exchanges on Tuesday, Aug. 20, were as follows: Francs, 3.91⅞c.; lire, 5.23c.; and marks, 23.815c. Canadian dollars, ⅝ per cent discount.

## Other Metals

**Quotations cover large wholesale lots unless otherwise specified**

ALUMINUM—Per lb., 99 per cent plus grades, price of leading interest, 24.30c. Outside market, 99 per cent plus, 24.30c.; 98-99 per cent, 23.90c.

ANTIMONY—Per lb.. duty paid. New York: Chinese brands, 8⅞c. per lb. for near-by positions, and 8¾c. for Aug.-Sept. shipment from China. Cookson's "C" grade, spot, 14c.

BISMUTH—Per lb., New York, in ton lots, $1.70. Smaller lots, $1.85 and up.

CADMIUM — Per lb.. New York: 80@90c. Active demand.

IRIDIUM — Per oz., $230@$240 for 98@99 per cent sponge and powder.

NICKEL—Per lb. ingot, 35c.; shot, 36c.; electrolytic, 35c. (99.9 per cent grade) for single lots of spot metal.

PALLADIUM — Per oz., $38@$40. Small lots bring up to $50. Nominal.

PLATINUM—Official price quoted by the leading interest on small miscellaneous orders for refined metal, $68 per oz. Transactions between dealers and refiners in the outside market are commonly reported at several dollars less.

QUICKSILVER — Per 76-lb. flask, $125@$126. Market holding firm, with additional business at $125 on round lots.

Prices of Chromium, Cobalt, Germanium Oxide, Lithium, Magnesium, Molybdenum, Osmium, Radium, Rhodium, Ruthenium, Selenium, Tantalum, Tellurium, Thallium, Titanium, Tungsten, Vanadium, and Zirconium are unchanged from the issue of Aug. 10.

## Metallic Ores

TUNGSTEN ORE—Per unit of $WO_3$, N. Y.: Wolframite, $15.75@$16.00 for future delivery. Withdrawals from bond have been heavy. Absence of news from China reflected in firm prices here. Bolivian scheelite is held at $16. Western scheelite quoted at $16.50 for October-November delivery.

Antimony, Chrome, Iron, Manganese, Tantalum, Titanium, Vanadium, and Zircon ores are unchanged from quotations in the Aug. 10 issue.

## Joplin Ore Prices Steady

*Joplin, Mo., Aug. 17, 1929*

**Blende**

| | | Per Ton |
|---|---|---|
| High | | $46.20 |
| Premium blende, basis 60% | | 44.00 |
| Prime Western, basis 60% | $43.00@ | 44.00 |
| Table concentrates, 60% | 41.00@ | 43.00 |
| Flotation concentrates, 60% | 39.00@ | 41.00 |
| Average settling price | | 43.92 |

**Galena**

| | |
|---|---|
| High | $86.90 |
| Basis 80% lead | 85.00 |
| Average settling price | 86.74 |

Shipments for the week: Blende, 12,538; lead, 1,638 tons. Value, all ores the week, $692,660.

A year ago slab-zinc price quotations in *Engineering and Mining Journal* were advanced from 6.200 to 6.250c. per pound and there has been no backward movement since, which may be accepted as evidence that no depressing stocks either of slab zinc or zinc ore have accumulated. The ore market situation is sufficiently firm that no changes in basis price quotations are reported.

For several weeks an increasing uneasiness was caused by a drought that threatened to close down several mills dependent upon stream water for wet concentration. Work was slowed up. A recent rain was insufficient to replenish fully the stream flow, but it is believed that rains will follow and at least maintain a low but constant supply. This tendency to slacken production will be augmented by one of the leading producing companies restricting its output to a five-day week beginning today, though no general movement will appear before the period following Labor Day, when a majority of the larger producing companies, it is intimated, will observe a week's holiday.

*Platteville, Wis., Aug. 17, 1929*

| Zinc Blende | Per Ton |
|---|---|
| Blende, basis 60 per cent zinc | $46.25 |
| **Lead Ore** | |
| Lead, basis 80 per cent | $85.00 |

Shipments for the week: Blende, 519 tons; lead, none. Shipments for the year: Blende, 21,785; lead, 1,180 tons. Shipments for the week to separating plants, 1,010 tons blende.

## Non-Metallic Minerals

Amblygonite, Asbestos, Barytes, Bauxite, Borax, Celestite, Chalk, China Clay, Diatomite, Emery, Feldspar, Fluorspar, Fuller's Earth, Garnet, Gilsonite, Graphite, Greensand, Gypsum, Iron Oxide, Lepidolite, Limestone, Magnesite, Mica, Monazite, Ocher, Phosphate, Potash, Pumice, Pyrites, Quartz Rock Crystal, Silica, Spodumene, Sulphur, Talc, and Tripoli are unchanged from prices in the Aug. 10 issue.

## Metallic Compounds

ARSENIOUS OXIDE (White Arsenic) Per lb., 4c. Good demand at steady prices; stocks diminishing.

SODIUM NITRATE — Per 100 lb., $2.07½ for spot deliveries.

Antimony Oxide, Calcium Molybdate, Copper Sulphate, Sodium Sulphate, and Zinc Oxide are unchanged from prices in the Aug. 10 issue.

## Alloys

FERROMANGANESE—Per long ton, domestic and foreign, f.o.b. furnace or duty paid, port of entry, $105. Spiegeleisen, 19@21 per cent, $34 f.o.b. furnace.

FERROTUNGSTEN—Per lb. of W contained, 75@80 per cent W, $1.45 f.o.b. works. Market steady.

Ferrocerium, Ferrochrome, Ferromolybdenum, Ferrophosphorus, Ferrosilicon, Ferrotitanium, Ferrotungsten, Ferrovanadium, Nickel Silver, and Yellow (Muntz) Metal are unchanged from prices in the Aug. 10 issue.

## Rolled Metals

Copper, Lead, Zinc, Monel Metal, and Nickel Sheets are unchanged from prices in the Aug. 10 issue.

## Refractories

Chrome Brick, Fireclay Brick, Magnesite Brick, Silica Brick and Zirkite are unchanged from quotations in the issue of Aug. 10.

## Iron, Steel, and Coke

IRON—Per gross ton, Valley furnaces, Bessemer, $19; basic, $18.50; No. 2 foundry, $18.50.

STEEL—Base prices per gross ton Pittsburgh, billets and slabs, $35; plates, structural shapes, and soft steel bars, per 100 lb., $1.95.

COKE—Per gross ton, Connellsville furnace, $2.75@$2.85. Connellsville foundry, $3.75@$4.85. Byproduct coke, Ohio and Kentucky (Connellsville basis) $6.50; Buffalo and Detroit, $8.50@$9.